VOLUME 104 PART A NUMBER 18 DECEMBER 1957



The Proceedings

OF

THE INSTITUTION OF ELECTRICAL ENGINEERS

FOUNDED 1871: INCORPORATED BY ROYAL CHARTER 1921

PART A

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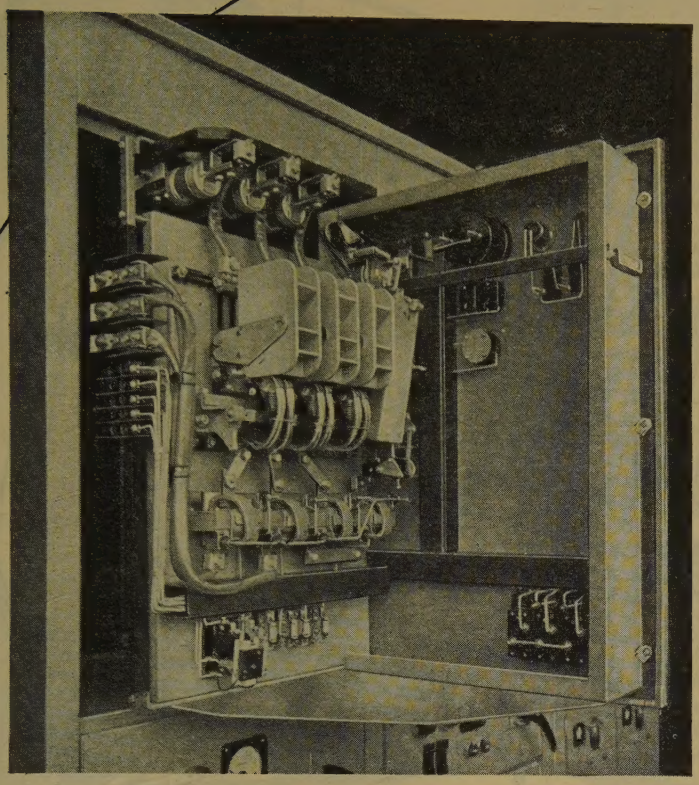
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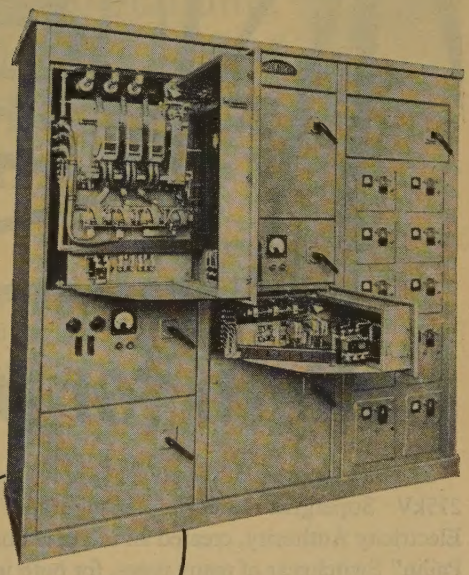
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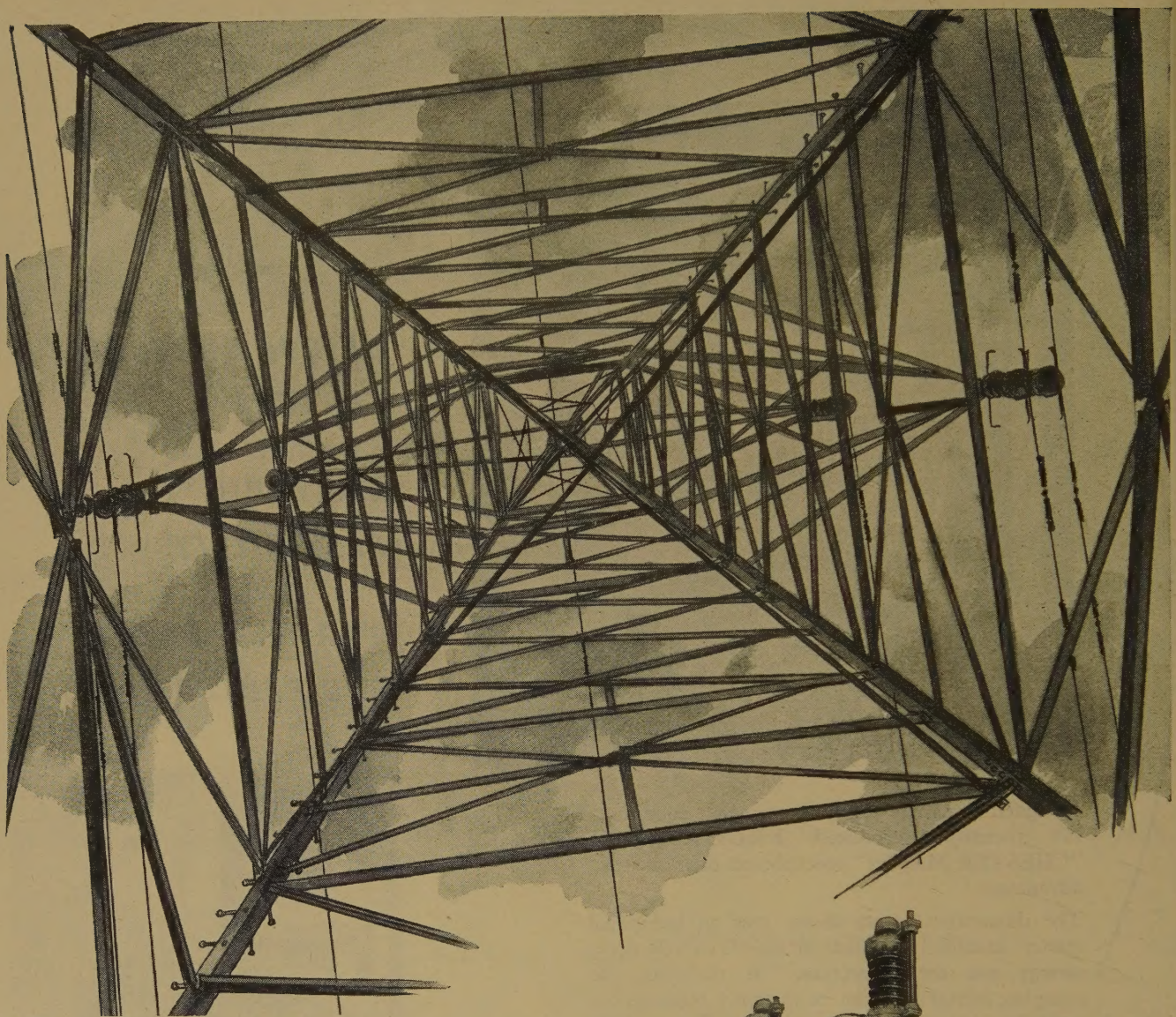
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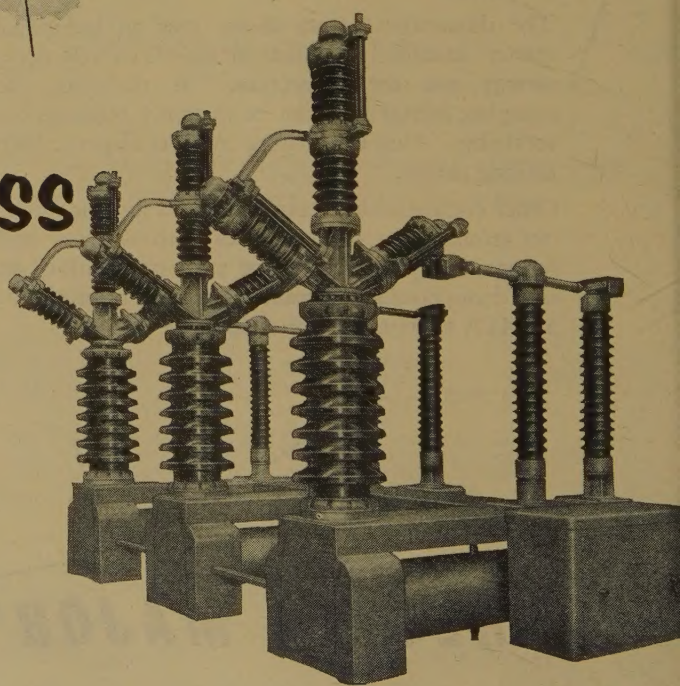
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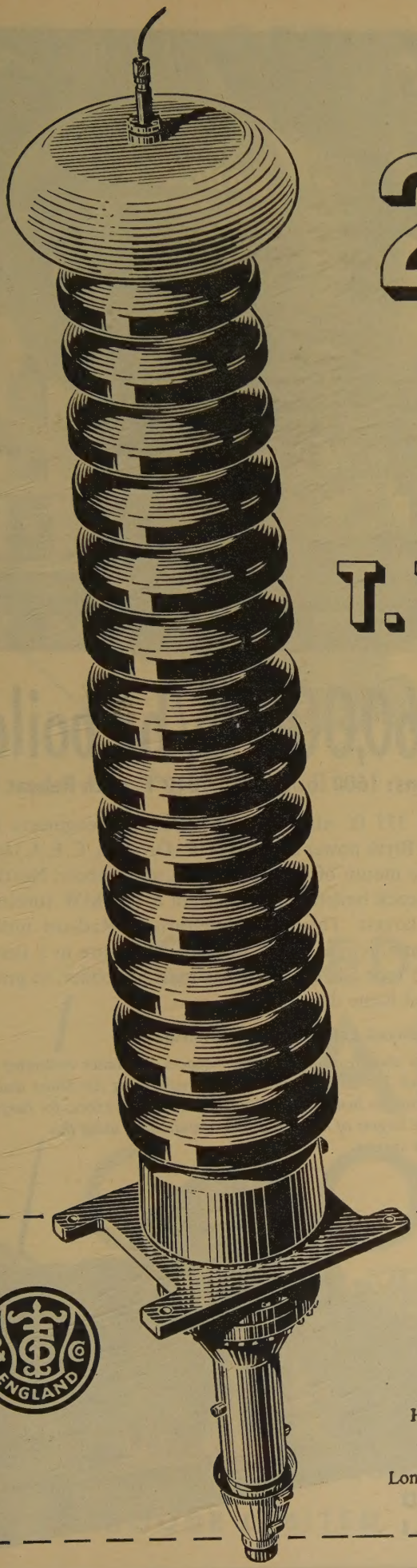
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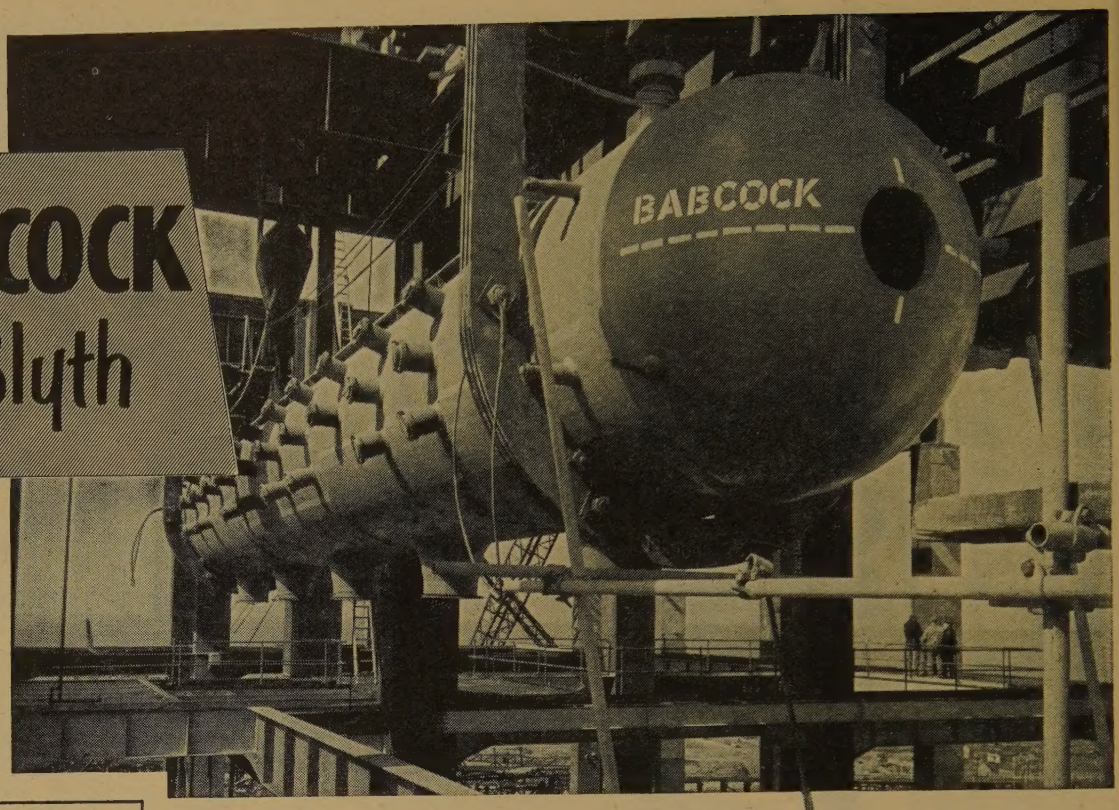
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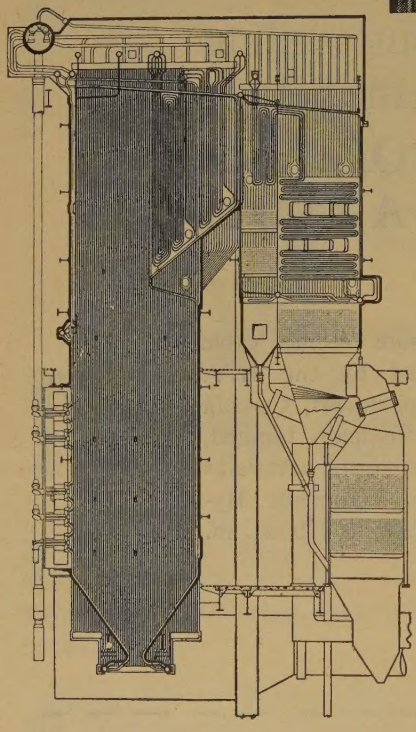
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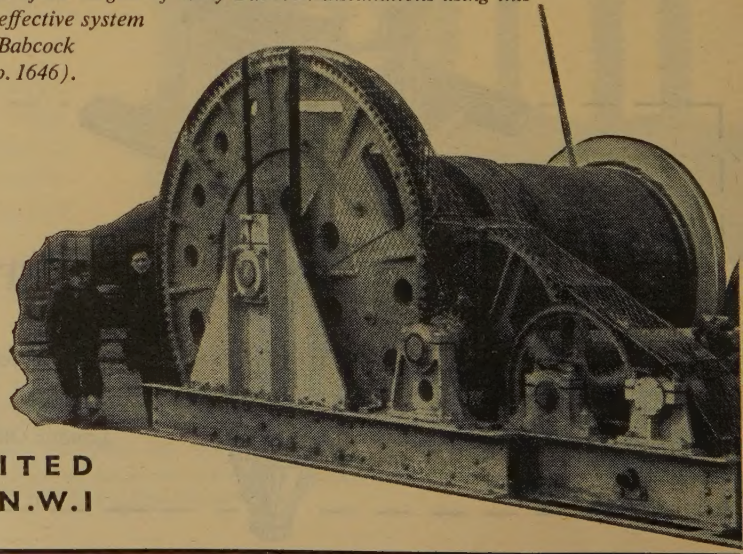
Steam Conditions: 1600 lb./sq. in., 1010°F. with Reheat to 1005°F.

From drum-level 137 ft. above ground, Babcock engineers erecting the boilers at the new Blyth power station (N. E. Division, C.E.A.) look out over the flat lands at the mouth of the River Blyth, at Cambois, Northumberland.

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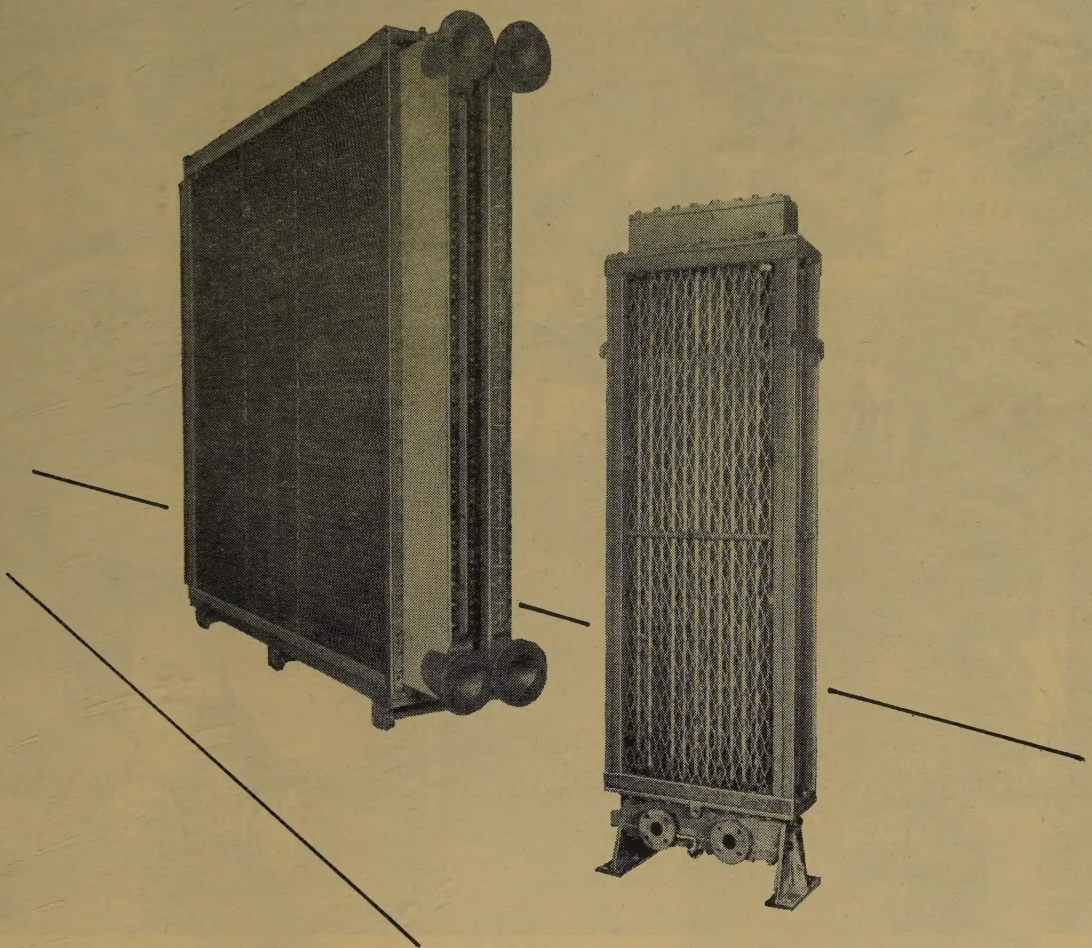
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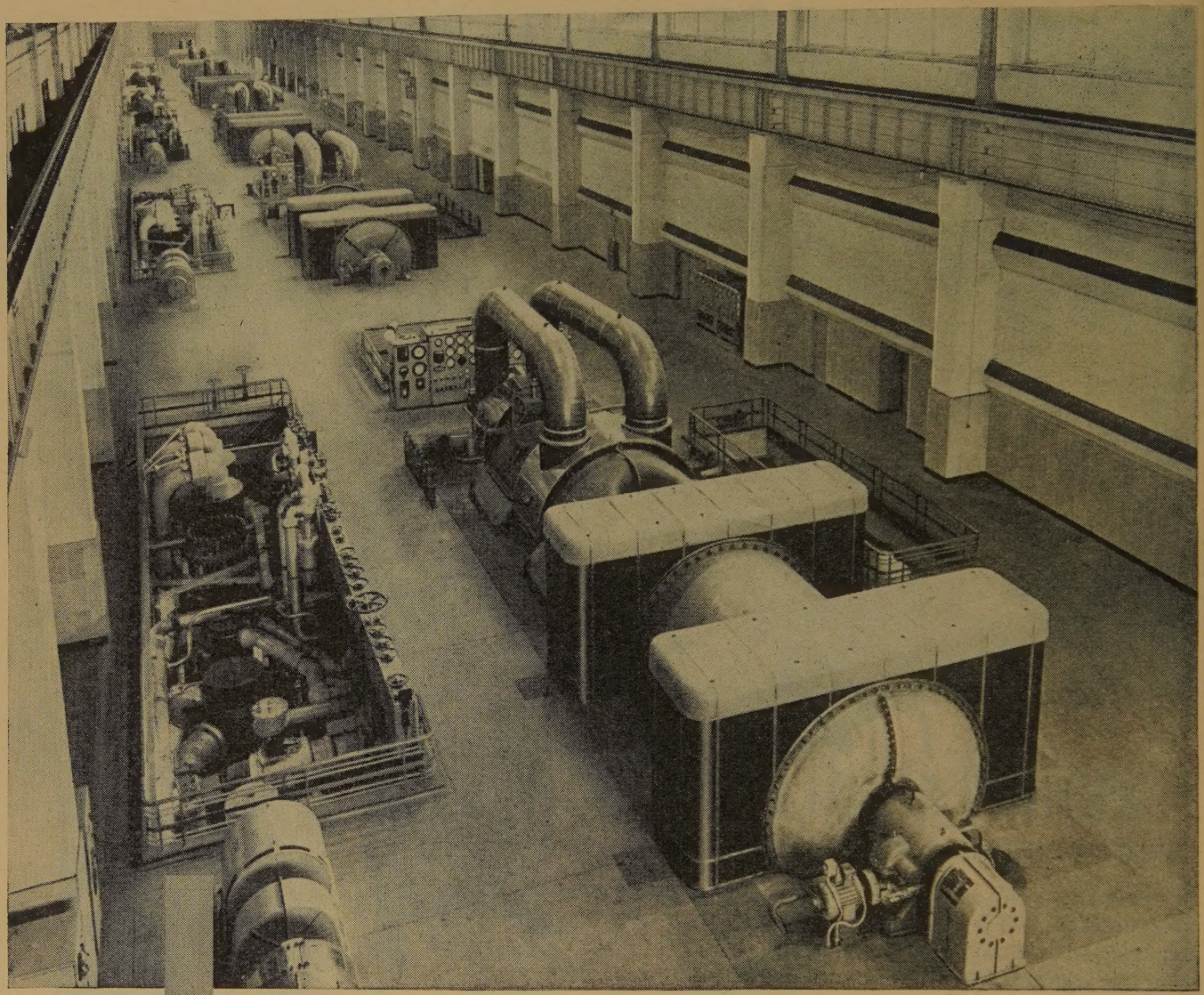


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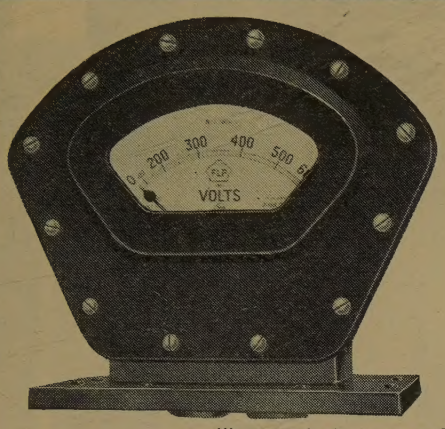
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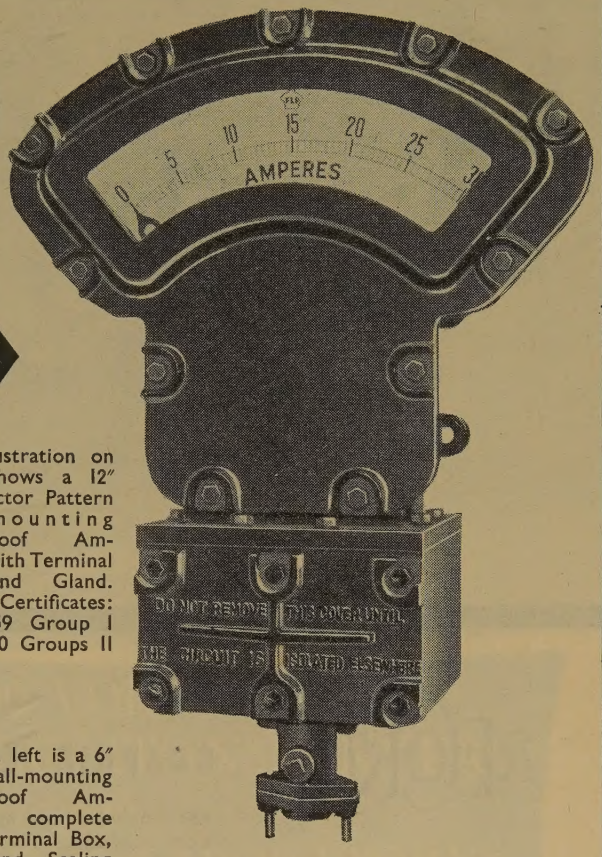
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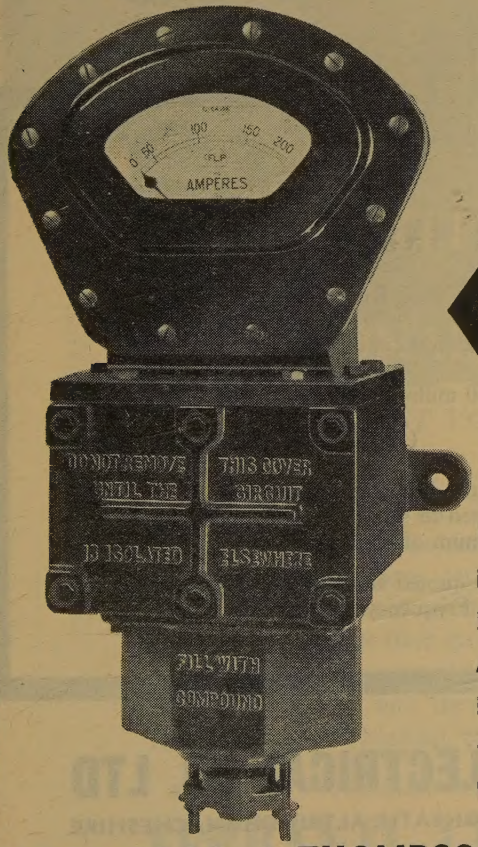
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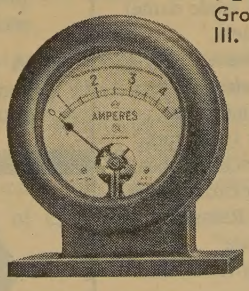


The illustration on right shows a 12″ dial, Sector Pattern Wall-mounting Flameproof Ammeter with Terminal Box and Gland. Buxton Certificates: FLP 3759 Group I FLP 3760 Groups II & III.



Seen on left is a 6″ dial, Wall-mounting Flameproof Ammeter, complete with Terminal Box, Compound Sealing Chamber and Gland. Buxton Certificates: FLP 1004/2 Groups I II & III.

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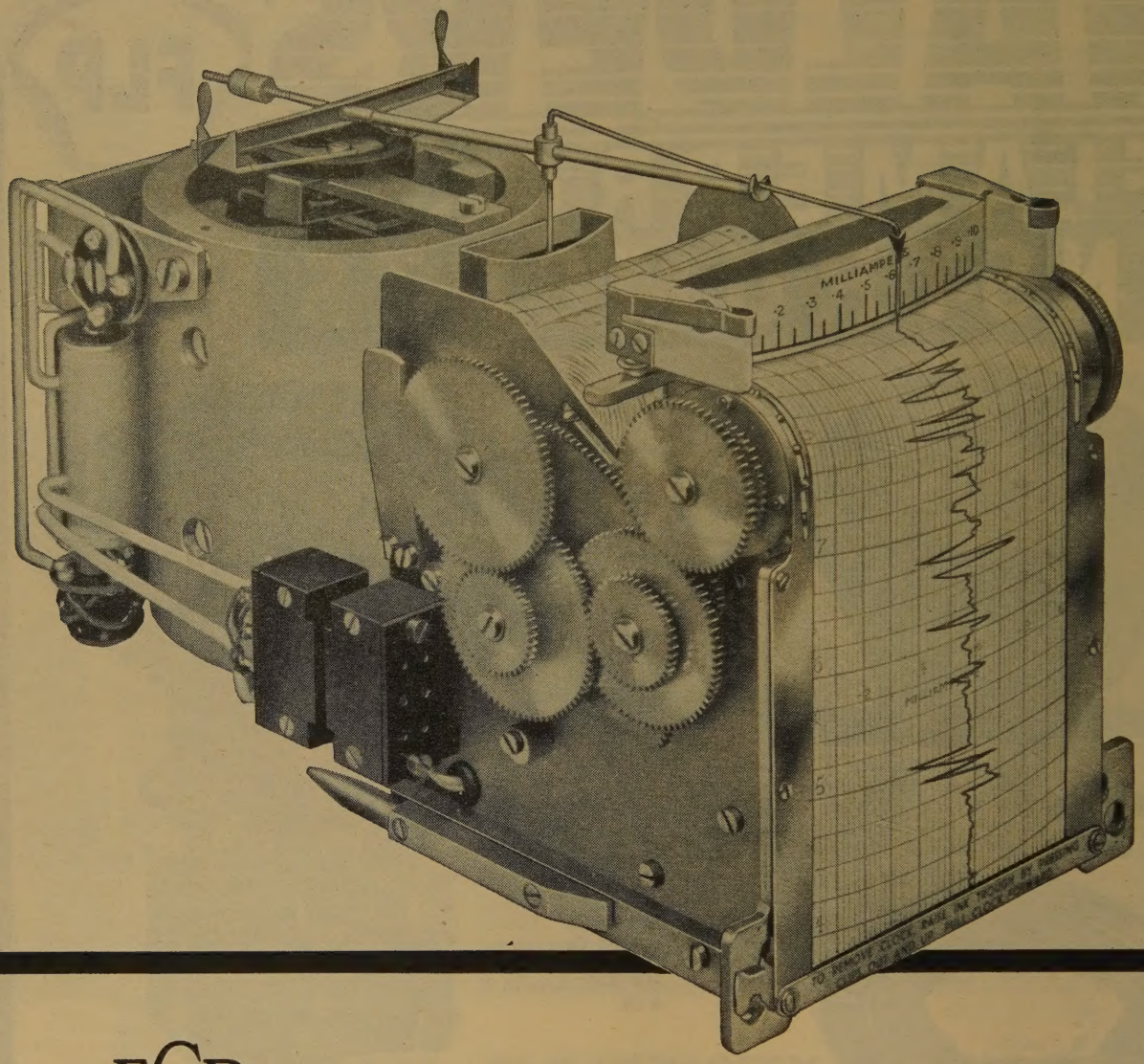
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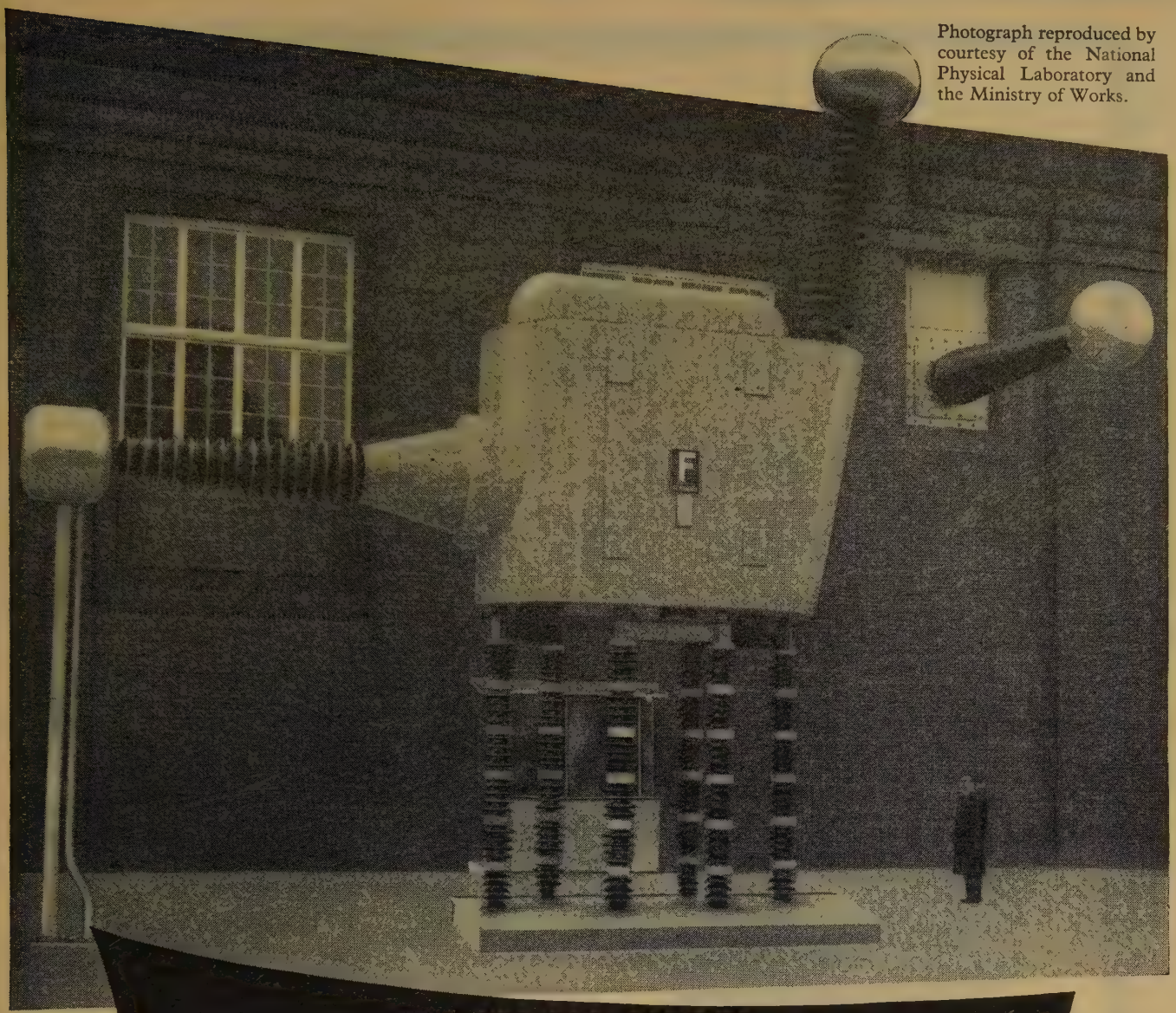
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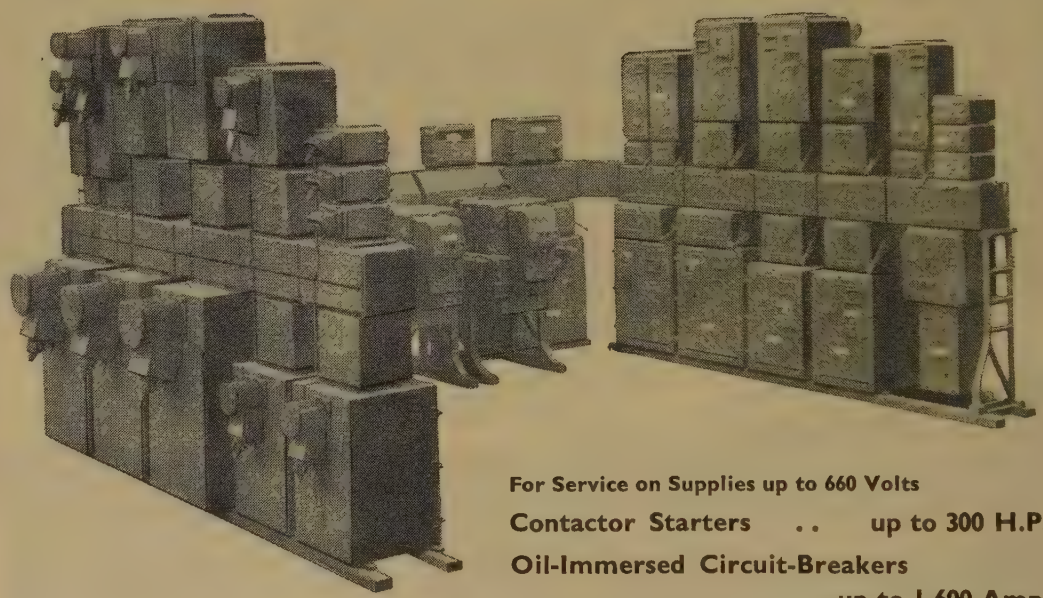
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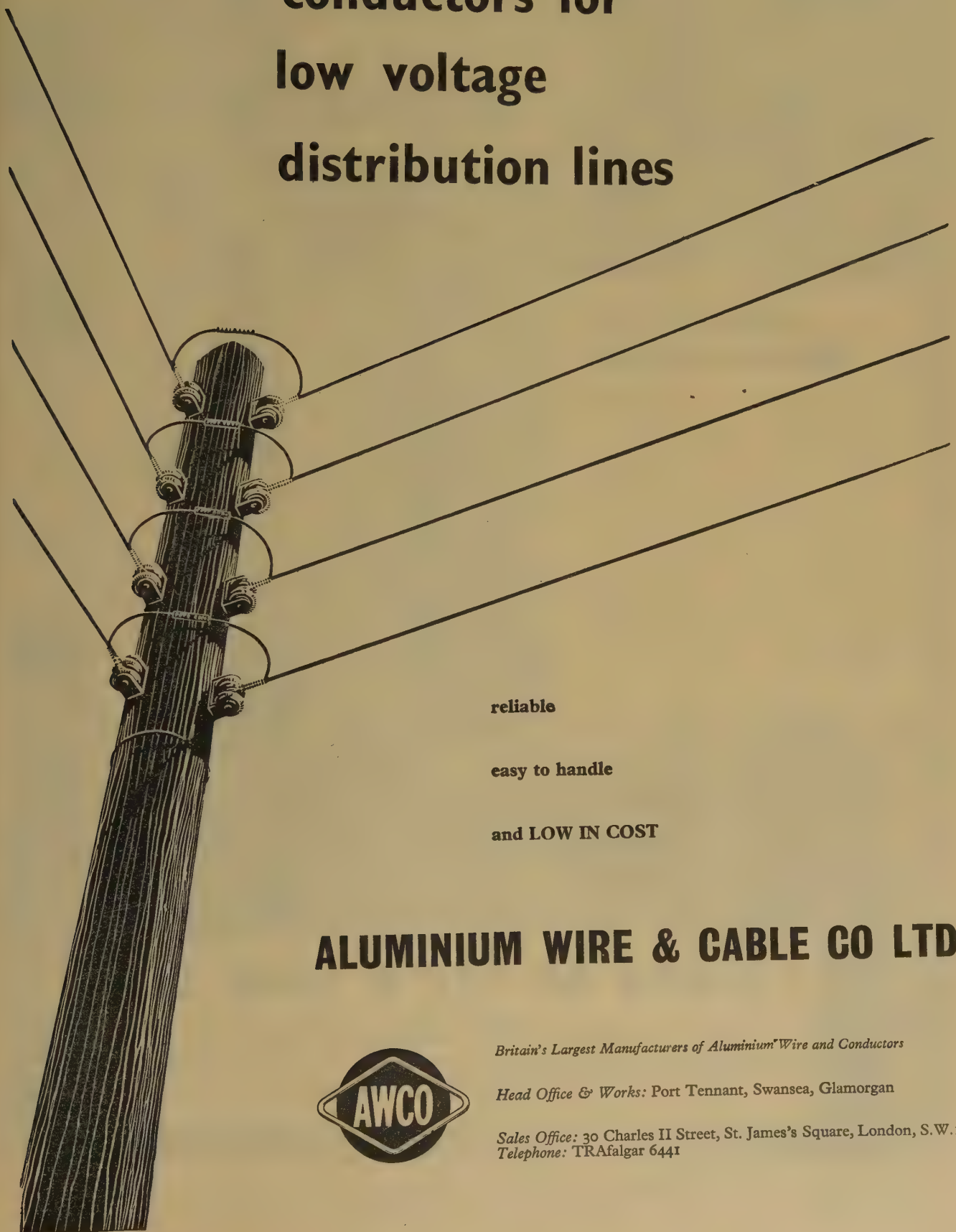
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AP 21



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are supplied up to 10″ wide x 1″ thick or 3″ square drawn finish; up to 12″ x ¾″ x 16′ hard rolled. Extra heavy machined bars to customers' requirements. Tubular and laminated bus-bars.

Over 70 tons of bus-bars were supplied for the installation in Murgatroyd's Salt & Chemical Company Limited, Sandbach, Cheshire. Illustrated are bars, 6″ wide x ¾″ thick, and bars 10″ wide by ¾″ thick were used in the main cell circuits.

The illustration shows copper connections on the switchgear floor for four 12,500 amp rectifiers: D.C. copper work on left; A.C. copper work on right. This installation was supplied by The English Electric Company Ltd., the Contractors being George E. Taylor & Co. (London) Ltd.

Pre-assembly work on bus-bars. Facilities are available in our Works for bending, sawing, drilling, slotting and tinning, to customers' requirements.

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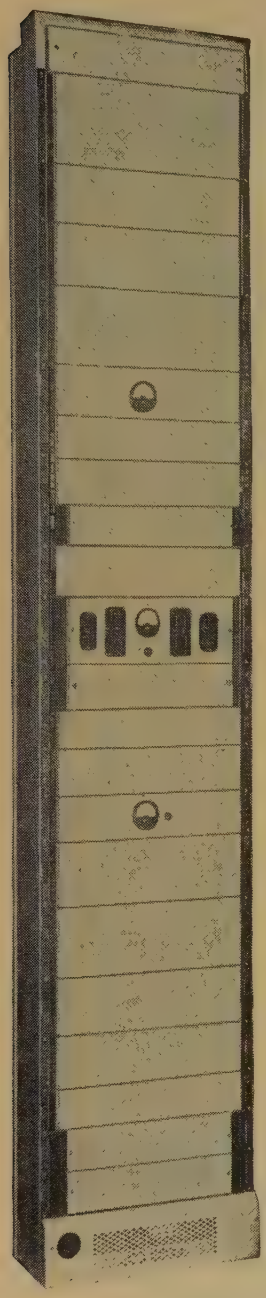
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- ★ **Crystal-controlled oscillator ensures effective tandem working.**
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- ★ **36 repeaters on one 9ft. rackside.**

FACILITIES

Rejection of start signal elements of a duration less than ⅛, ¼, ⅜ or ½ of a signal element, as required.

Automatic insertion of stop signal element in the retransmitted code regardless of its presence or not in the received signal.

Automatic re-synchronisation after the regeneration of each signal.

The minimum signal length of 138.125 ms (at 50 Bauds) allows for the transmission of both 7 and 7½ unit code signals, in accordance with C.C.I.T.T.

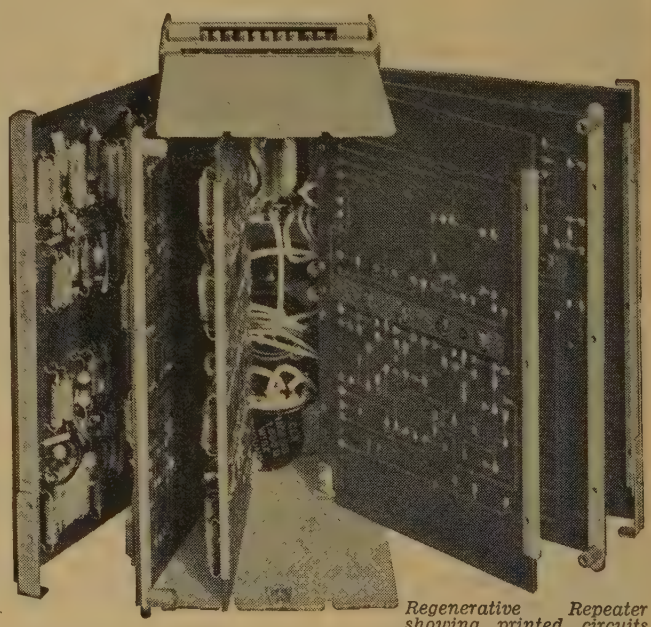
The repeaters can be made to operate at signalling speeds of up to 100 Bauds.

The degree of distortion experienced by start-stop telegraph signals during transmission can be so great as to render them unintelligible to a teleprinter.

Such distortion can occur in radio links, in switched telegraph networks employing a number of tandem connections and on land lines where long distances separate the terminals.

Standard Transistorised Regenerative Repeater Equipment will recreate, or reform, telegraph signals with zero distortion for retransmission either directly into a receiving teleprinter or over a further telegraph link.

The equipment will receive, examine and regenerate start-stop telegraph signals which may be suffering distortion of up to 49%. Distortion of the signal transmitted by the equipment never exceeds 1%.



Regenerative Repeater showing printed circuits and component assemblies

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Germanium Power Rectifiers

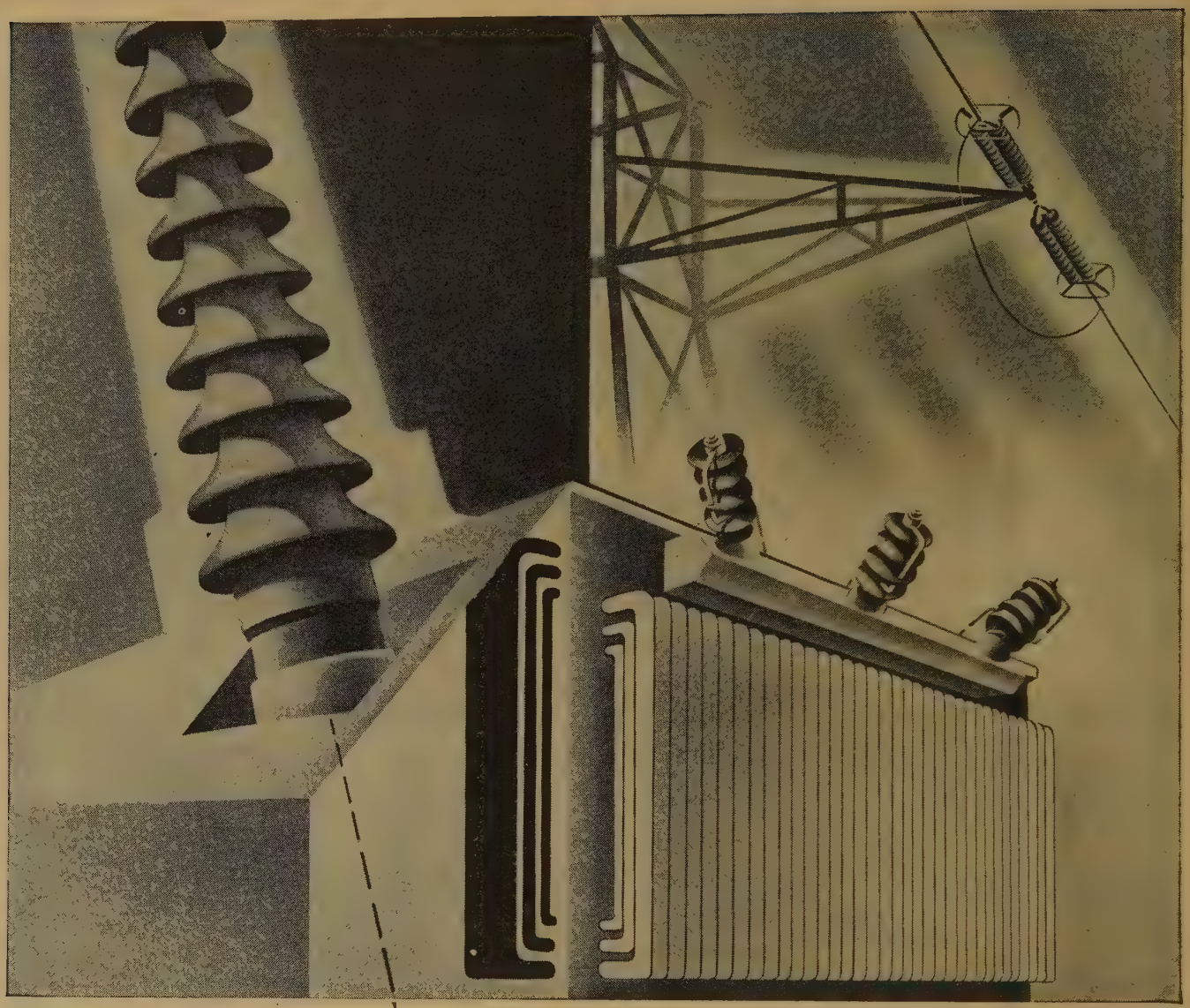
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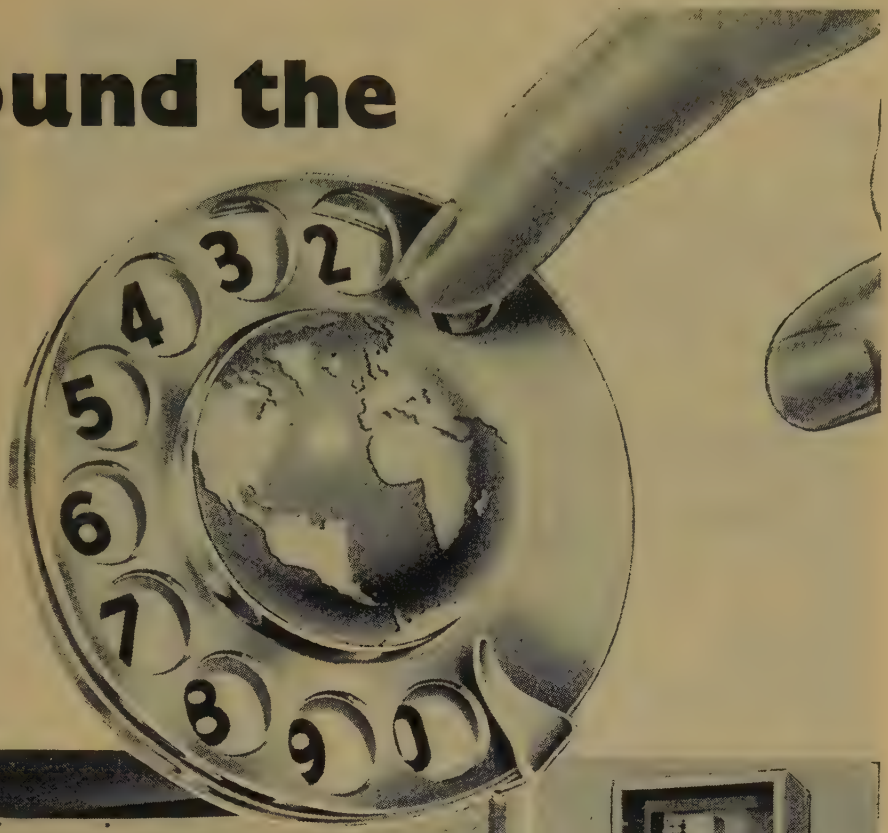
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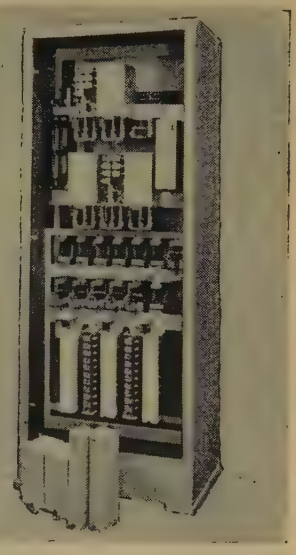
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G.E.C. telecommunication systems bridge the seas, span deserts, level mountain ranges. If you are planning a comprehensive network, or want a small inter-communication system, let G.E.C. tell you of the wide range of equipment available.







A typical G.E.C. remote supervisory control room, with (right) a typical private automatic exchange unit. G.E.C. remote supervisory control provides the power engineer with the means of controlling power distribution from a central control room, and with speech facilities, long-range meter readings, and state-of-switchgear indications.



Left:—Line coupling equipment through which carrier signals are injected into the power lines themselves. Up to eight communication circuits can be provided by each system.

G.E.C.

THE GENERAL ELECTRIC CO. LTD. OF ENGLAND

TELEPHONE, RADIO AND TELEVISION WORKS, COVENTRY





Top: Seven-panel switchboard on a restricted site in South Wales.
Left: Single unit with tank lowered and breaker in isolated position.

ONE TENTH THE SPACE

The UE1 air insulated 33-kV outdoor metalclad switchboard, the first of its kind produced in Britain, has been designed to meet the growing needs of 33-kV networks, and in particular to overcome the problem of space limitation as well as providing economy and ease of maintenance. An outdoor substation composed of seven UE1 units occupies one tenth the space (900 sq. ft. compared with the 9,000 sq. ft.) required by a conventional unit. A motor-operated mechanism is incorporated for raising and lowering the oil circuit breaker, thus providing for full inspection under cover in all weather conditions. For further details please write for publication No. 542.

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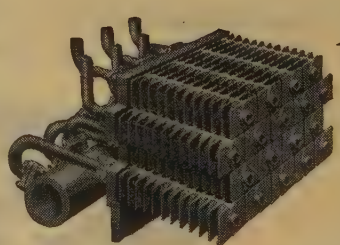
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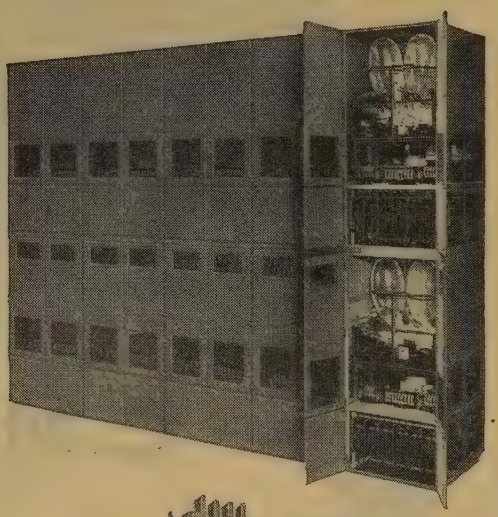
60,000 amps

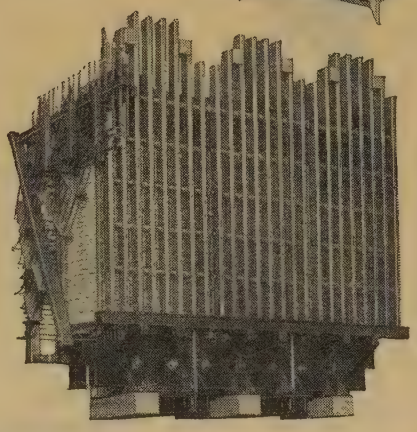
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in one installation

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handle any capacity





This record size installation of Hewittic rectifiers provides a continuous D.C. output for a large electro chemical plant. The same skill and attention to detail is devoted to the manufacture of all types of Hewittic rectifiers, and this equipment—by far the largest glass bulb rectifier installation in the world—serves to emphasise the supreme reliability and high efficiency of the Hewittic rectifier and its complete suitability for handling any capacity that may be required.

The photographs on the right show, *top*, one of the banks of Hewittic rectifiers depicted above and, *bottom*, a view from the secondary side of one of the main transformers for this installation.

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CMA 16



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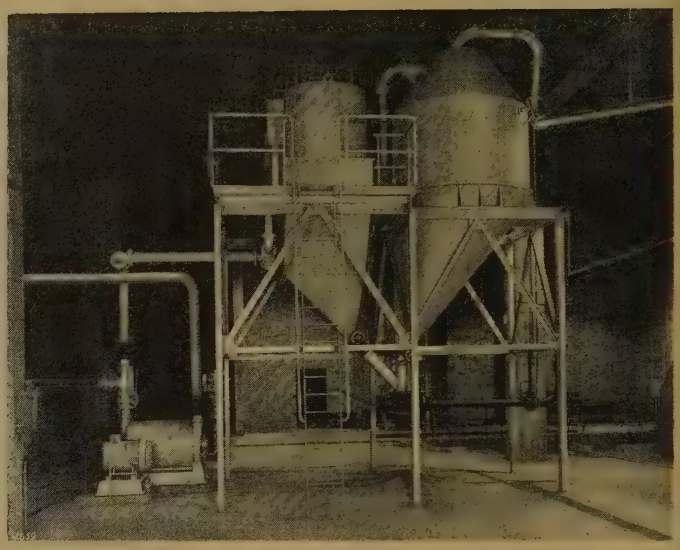
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Sturtevant Turbo-Exhausters create the very powerful suction required, and the dust collection and settling apparatus in its various forms deals efficiently with all types of dust.

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- **Surface cleaning.**
- **Emptying the ash pit under a stoker-fired boiler.**
- **Cleaning a pulveriser.**

Please write to our reference N/101/EB for further particulars.

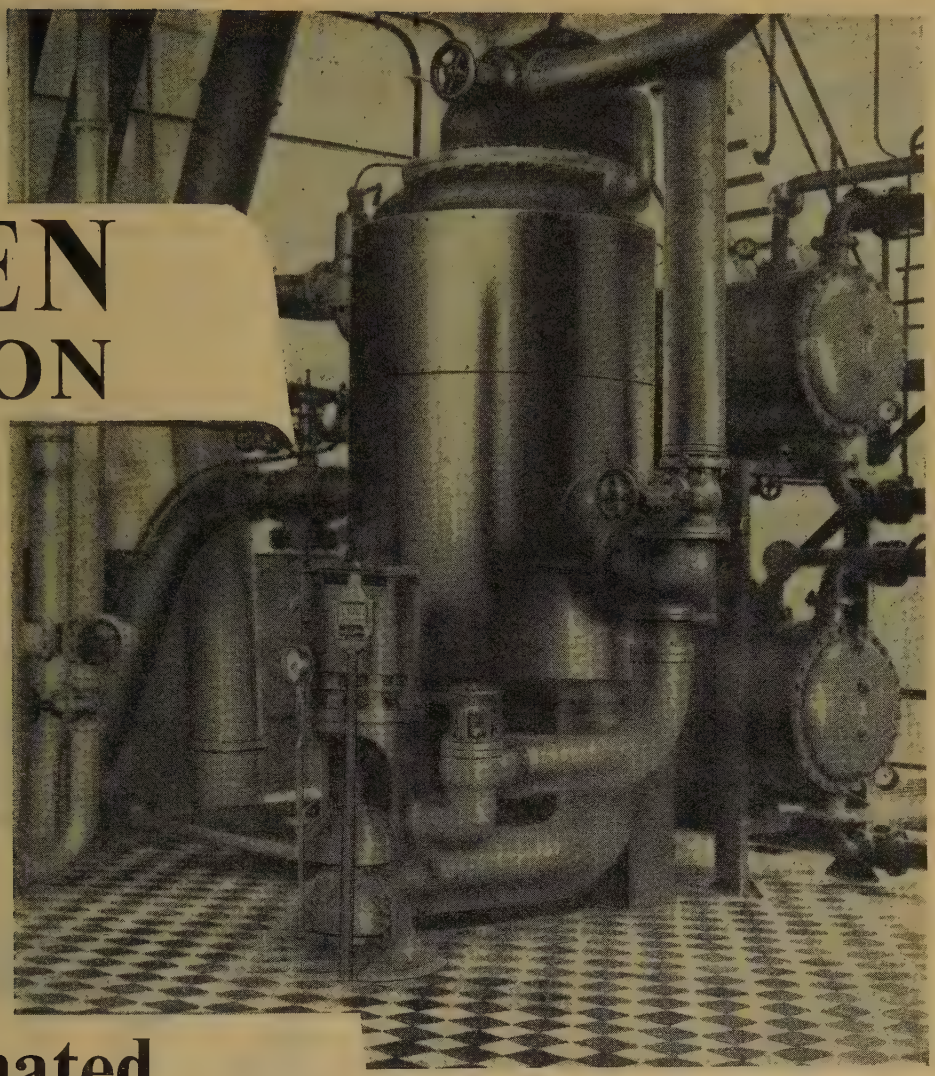




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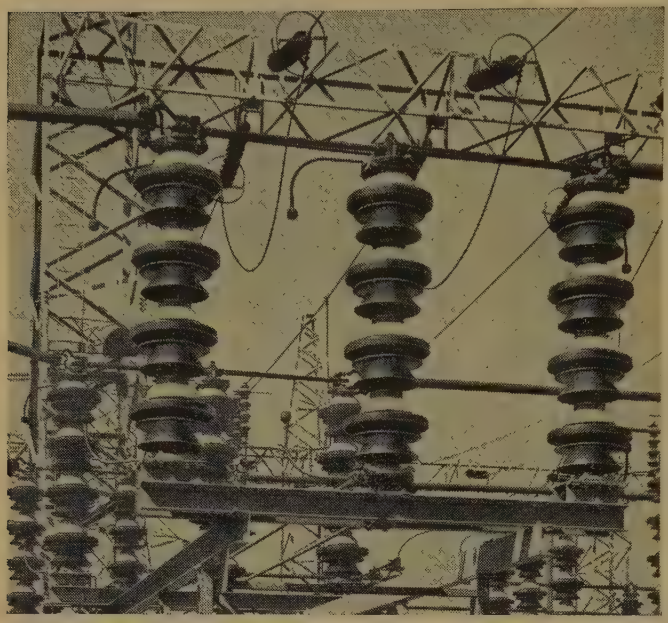
WITH THE WEIR "OPTIMUM" DE-AERATOR which is the simplest and most effective equipment for removing corrosive gases dissolved in boiler feed water, and so protecting boiler surfaces, piping, economisers, etc., from corrosion and pitting. The Weir De-aerator can readily be installed in existing power plants, and heat in the operating steam is retained in the feed water.



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POST INSULATORS made by Doulton for voltages up to 132 kV are in use throughout the world. Apart from the large quantities of units supplied to switchgear manufacturers in England which ultimately go overseas, many thousands of Doulton Post Insulators are shipped annually direct to Australia, India, South Africa, Canada, New Zealand, Eire and other countries abroad now in the process of electrical development. The illustration shows Doulton heavy duty Post Insulators in 4-stack units in isolating switches and busbar supports on a 132 kV sub-station in New South Wales.

Photograph by courtesy of
The Electricity Commission of New South Wales.

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Simon-Carves electro-precipitators at the National Coal Board's "Phurnacite" plant at Aberaman

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This is believed to be the highest efficiency ever recorded in this country for a dry-type coal-dust precipitator. It was achieved under overload conditions, six months after commissioning.

High efficiency electro-precipitation by **Simon-Carves Ltd**

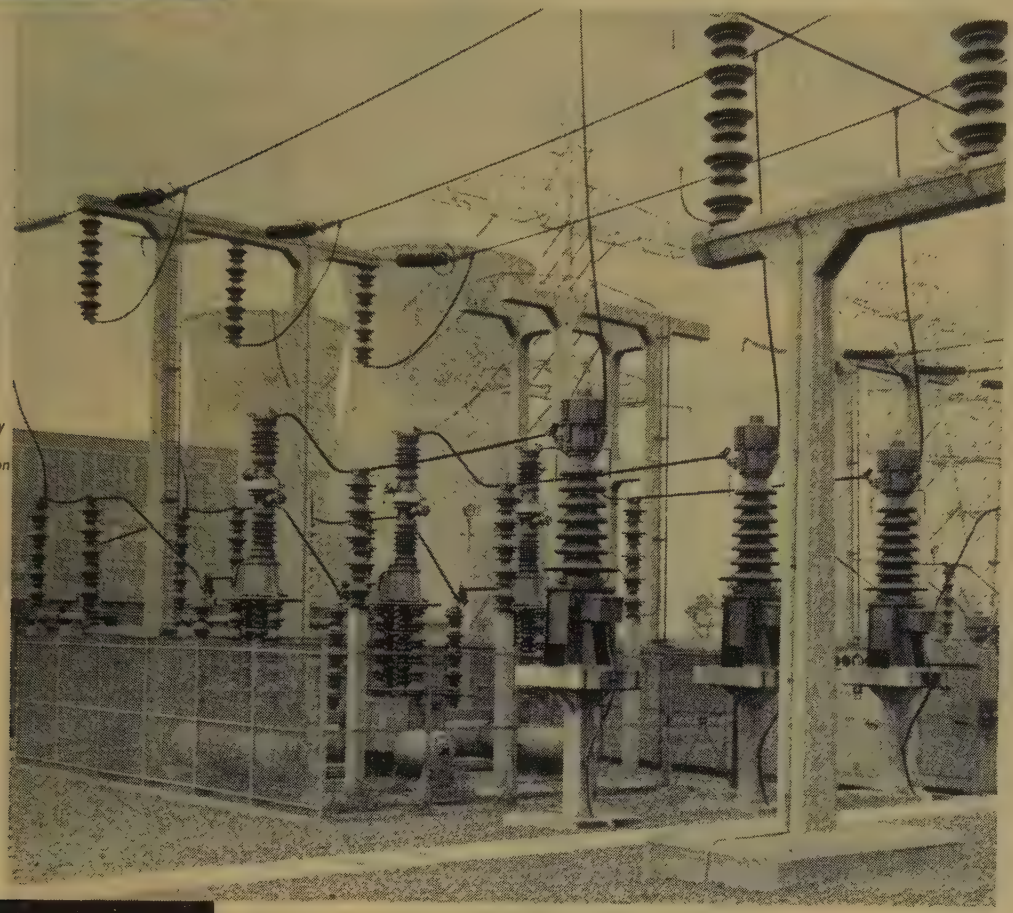
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SC 176

WAKEFIELD B POWER STATION



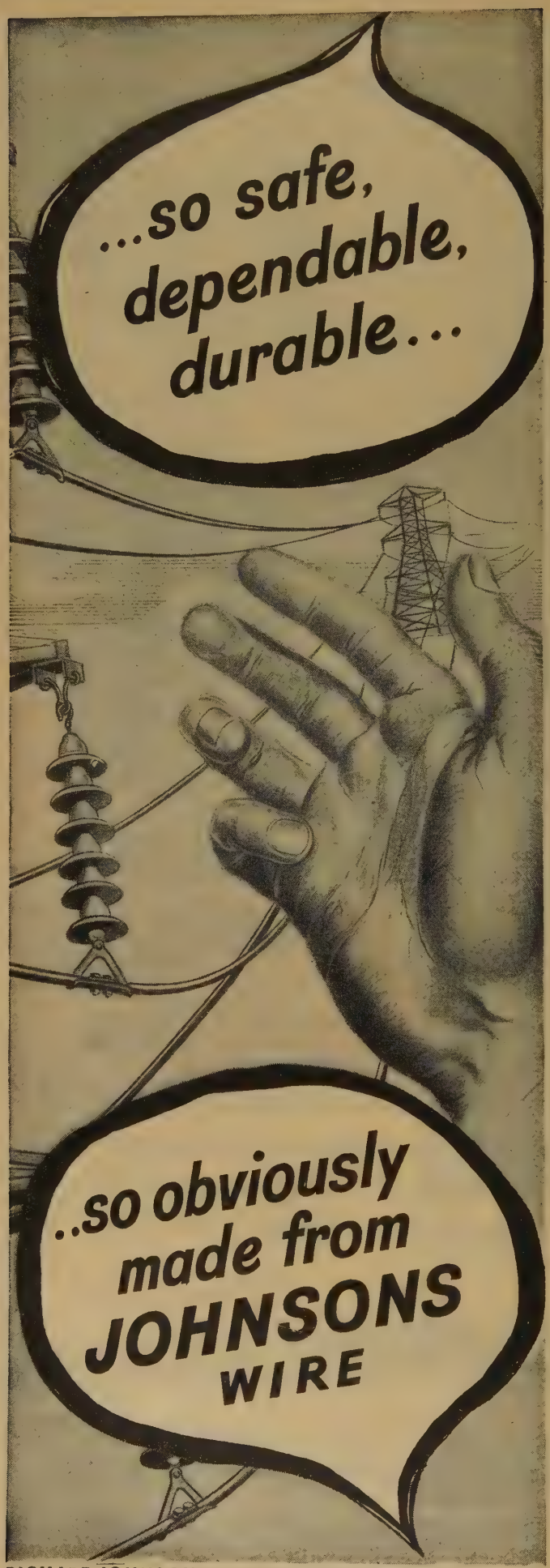
Photographed by courtesy of C.E.A., Yorkshire Division

REYROLLE SWITCHGEAR

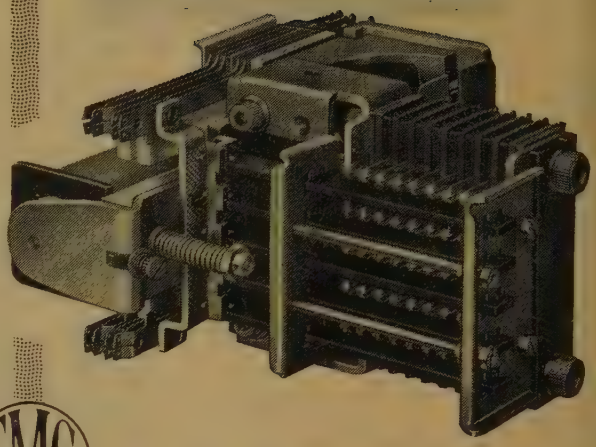
The main 132-kV 2,500-MVA air-blast switchgear at Wakefield B Power Station together with its associated control equipment, was supplied by Reyrolle who were also responsible for the 3·3-kV auxiliary air-break switchgear

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Multi-point Switching Unit

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The Bridge comprises 5 banks of 8 make-contacts. Used as a single unit it is capable of selecting any one of 5 points. In conjunction with a simple change-over relay it can select any one of 10 points (4 way). Two or more Bridges can be arranged to interact electrically to provide complex counting or stepping facilities. Moreover, the Bridge overcomes those problems associated with rotary wiper-contact selecting mechanisms which, in operation, are liable to generate spurious noise.

All contacts, which are silver, are duplicated on either side of the selecting card, and robust construction ensures that the stability of the contact gaps is such that readjustment should not be necessary during the normal life of the unit.

When your project involves multi-point switching, the TMC CROSSBAR BRIDGE may well be the solution to the problem. *Ask us to send you the full technical details.*

Designed and Manufactured by:

TELEPHONE MANUFACTURING CO LTD

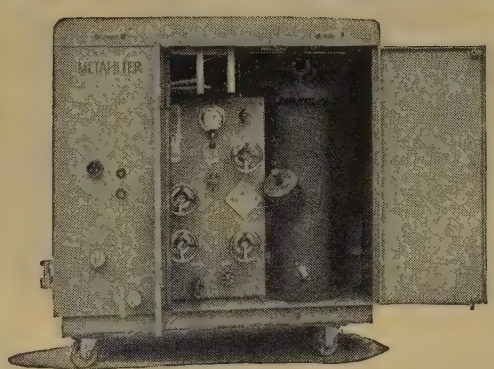
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Metafilters for transformer and switch oils are available as static, portable and fully mobile units with capacities from 50 g.p.h. to 1000 g.p.h. Our experience and advice is at your disposal. Please write for Section 7 Catalogue giving details of our latest units.

Metafilter

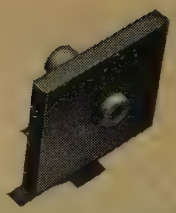
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To illustrate the extreme hardness of SINTOX Industrial Ceramic, the photograph on the right shows a ½ in. diameter tube of SINTOX which, without even chipping, was forced through a ¼ in. thick mild steel plate.



SINTOX IN THE ELECTRICAL INDUSTRY



Compared to porcelain, in tension, compression and cross breaking, SINTOX has at least twice the strength; and insulating properties twice as good.

Electrical resistance—SINTOX is so little affected by temperatures that at 300° C. it is about one million times that of porcelain. Thermal conductivity twenty times better than porcelain—almost comparable with steel.

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THE HARD CERAMIC **SINTOX**

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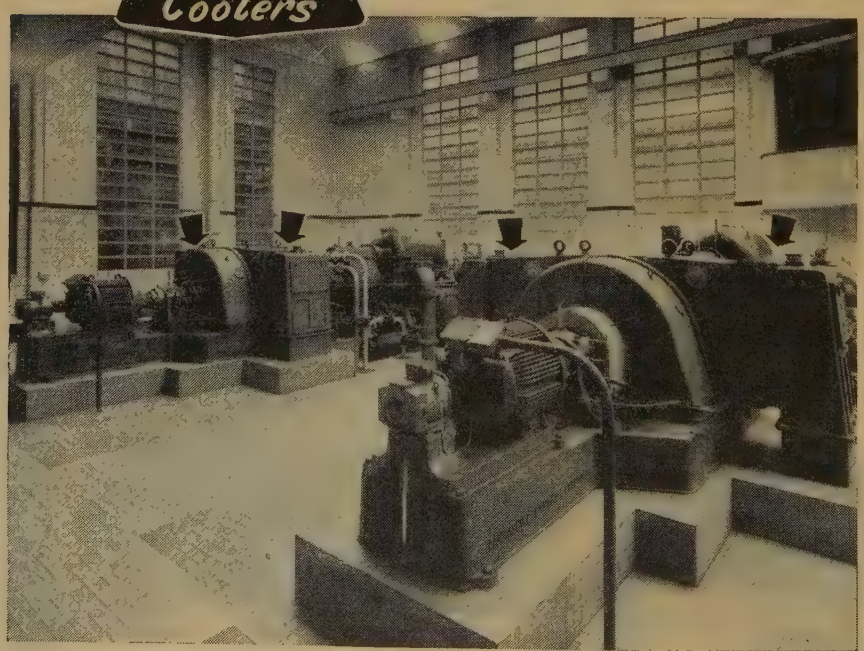
Sintox Technical Advisory Service

This service is freely available without obligation to those requiring technical advice on the application of Sintox Industrial Ceramics. Please write for booklet or any information required enclosing blue print if available.

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for ALTERNATORS GENERATORS TRANSFORMERS VALVES and RECTIFIERS



Closed circuit Spiral Tube Air Coolers fitted to Alternators manufactured by the Lancashire Dynamo & Crypto Ltd., installed at the East Greenwich Power House the of S.E. Gas Board.

Over many years the company's Technicians have progressively developed special cooling equipment in conjunction with Alternator Motor and Transformer Manufacturers.

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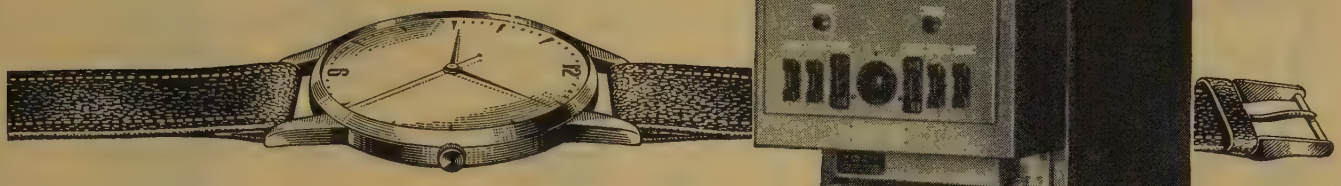
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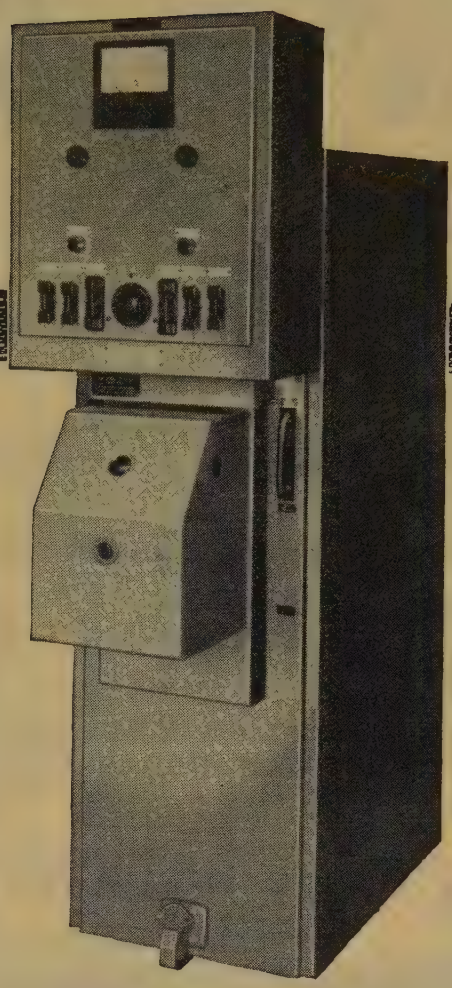
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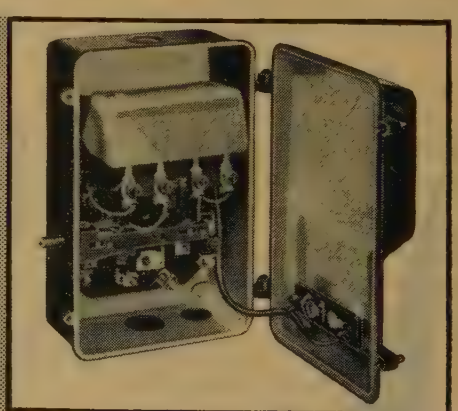
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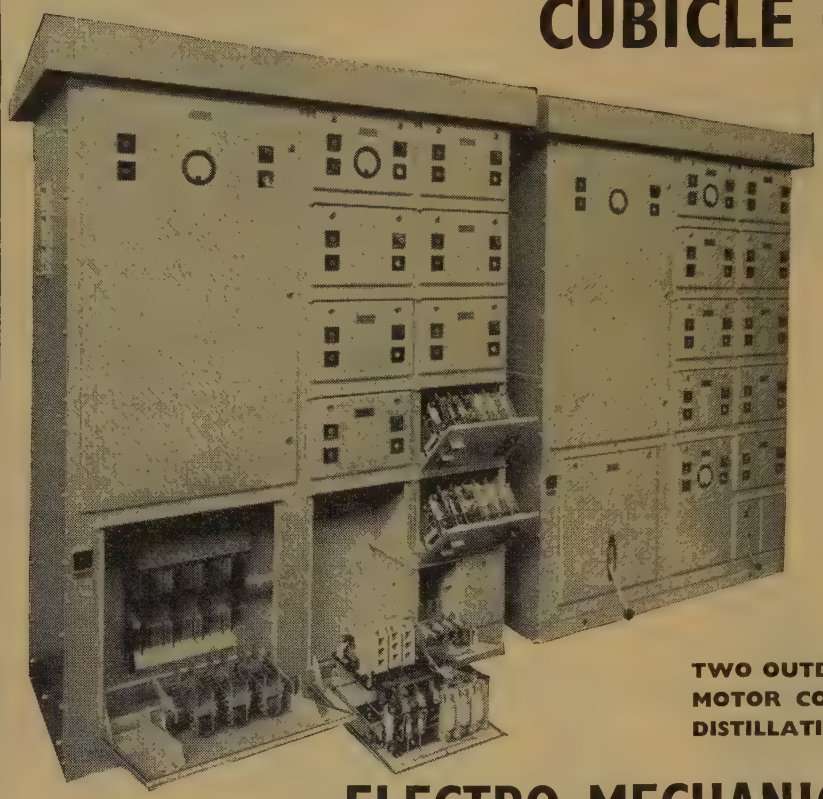
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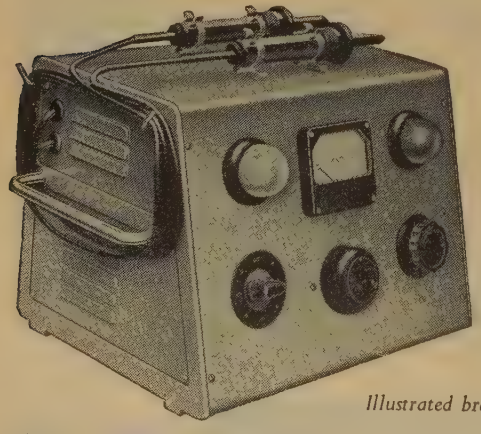
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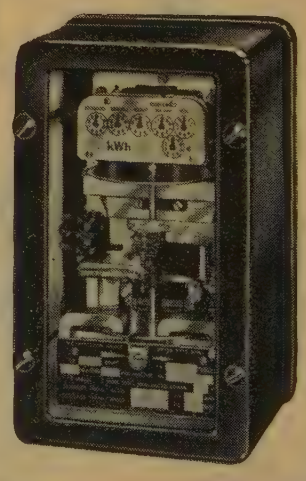
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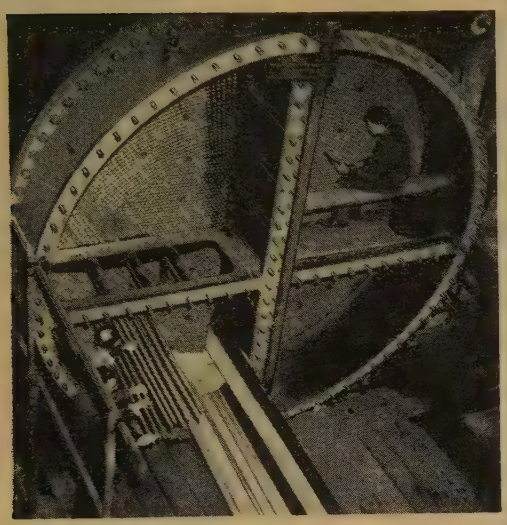
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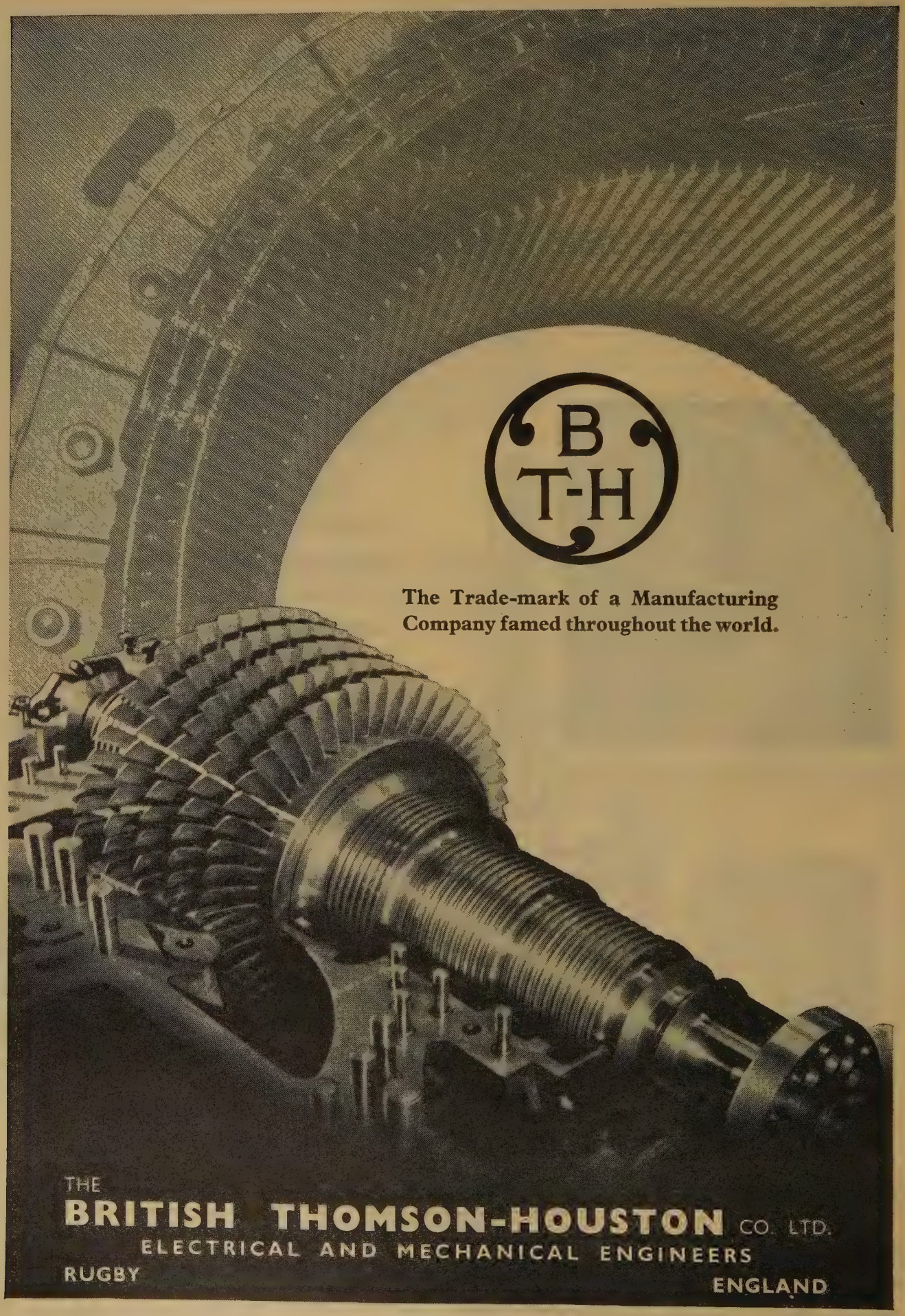
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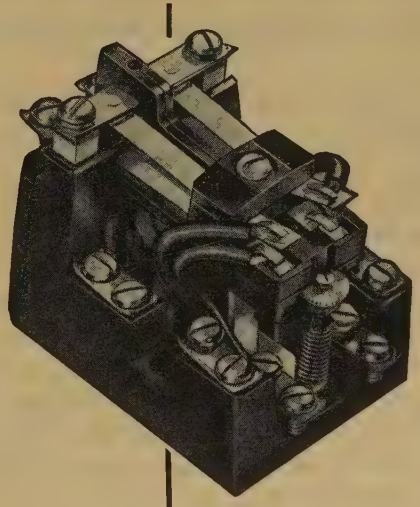
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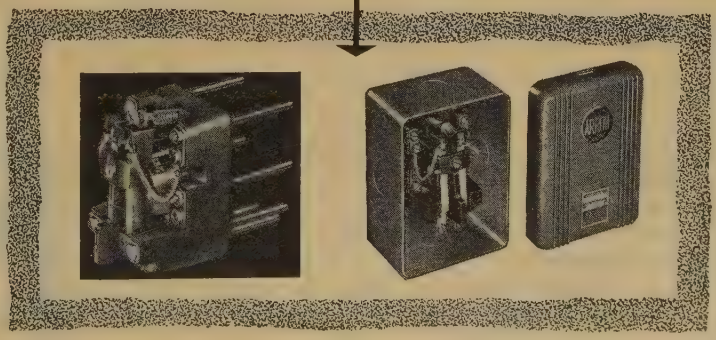
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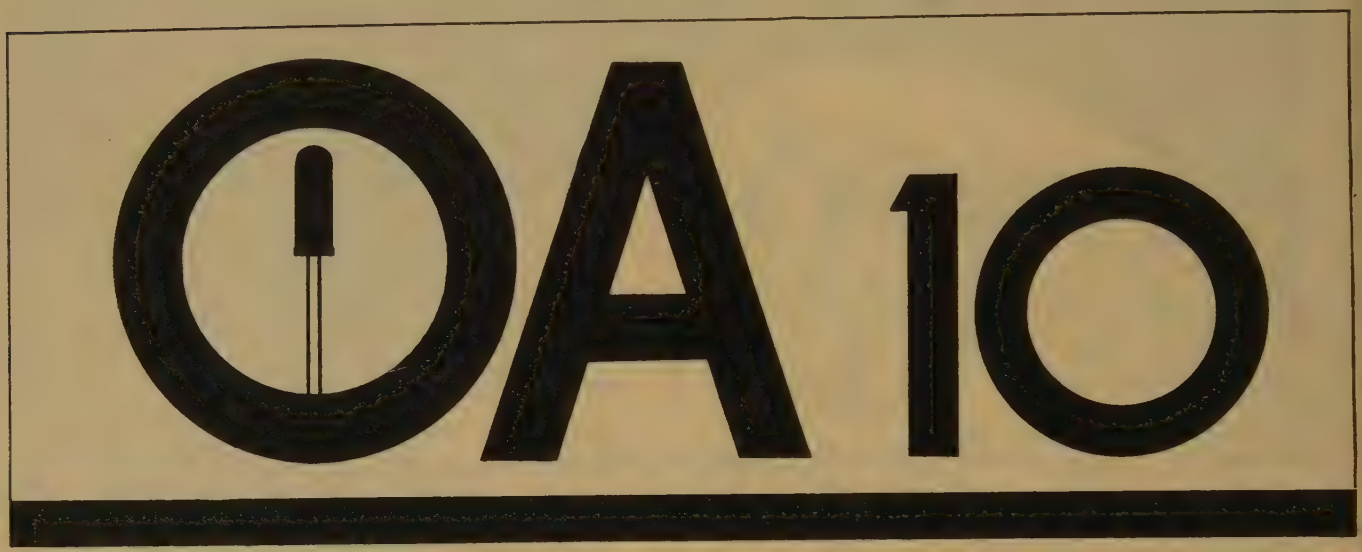


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Peak (5μsec max.)	(A)	1	1
Average (50msec averaging time)	(mA)	100	100
Average Characteristics			
Forward voltage (at 0.3mA)	(V)	0.2	0.13
„ „ (at 100mA)	(V)	0.5	0.44
Reverse leakage current (at −20V)	(μA)	1.5	10
Hole storage recovery time*	(μsec)	0.35	0.35
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* For test conditions see published data

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VOL. 104. PART A. No. 18. DECEMBER 1957 ©

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The Institution of Electrical Engineers
Paper No. 2399 U
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A 2 : 1 POLE-CHANGING INDUCTION MOTOR OF IMPROVED PERFORMANCE

By Prof. G. H. RAWCLIFFE, M.A., D.Sc., Member, and R. F. BURBIDGE, B.Sc., Graduate.

(*The paper was first received* 17*th January, and in revised form* 3*rd June*, 1957.)

SUMMARY

The paper explains a very simple method of much improving the performance of 2 : 1 pole-changing induction motors. In spite of the extensive use of such machines for a very long time past, windings of standard type, as described in most advanced textbooks, seem always to have been used. The new winding discussed in the paper has given excellent results on test.

(1) INTRODUCTION

There are two very well known methods of obtaining a speed change in induction motors, in the ratio 2 : 1, by the method of pole changing. In one of these methods [see Fig. 1(*a*)], the

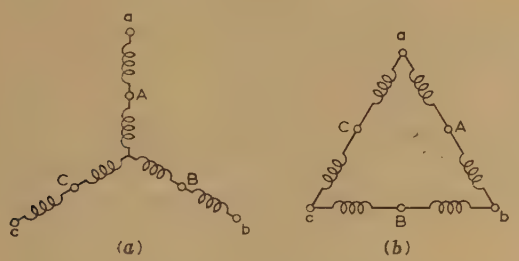


Fig. 1.—Standard 2 : 1 pole-changing windings.

(*a*) Parallel-star/series-star
p poles/2*p* poles
(*b*) Parallel-star/series-delta
p poles/2*p* poles

p poles	Supply	A B C	⊙ Six control leads
	Join	*a b c*	
2*p* poles	Supply	*a b c*	
	Disconnect	A B C	

stator windings are connected in series-star at the lower speed, and in parallel-star at the higher speed. In the other method [see Fig. 1(*b*)], the stator windings are connected in series-delta at the lower speed, and in parallel-star at the higher speed.

It is also possible to use star connection at the lower speed, and either star or delta connection at the higher speed, though—for reasons which will appear—these variants are less used than the first two mentioned above. An essential feature of all these methods of 2 : 1 pole-changing, however, is that parallel-star connection shall be used at either one speed or the other; and the modification* proposed in the paper in every case consists in adding some form of auxiliary winding to the parallel-star connection, in order to equalize the working fluxes at the two speeds.

(2) DISADVANTAGES IN EXISTING PRACTICE

The principal disadvantage in all these earlier methods is that the machine does not operate at full air-gap flux density at both speeds. In the first method, the flux density at the lower speed is too low when the machine is wound to give full flux density at the higher speed. In the second method, the flux density is too low at the higher speed when the machine is wound to give full flux density at the lower speed. In each of these two methods, therefore, the continuous power rating at one speed or the other is much less than the normal rating of the machine for that speed alone, and the corresponding pull-out torque is much lower than would be given by a normal single-speed winding in the same frame.

The other two methods of connection shown in Table 1 give an even greater disproportion between the air-gap flux densities at the two speeds; and it is for this reason that they are less used, though they have found occasional employment, especially the last method.

(3) PRINCIPLE OF NEW WINDING

In all these four methods of 2 : 1 pole changing only six leads (A, B, C, *a*, *b*, *c*) have to be brought from the winding to the terminal board, as shown in Figs. 1(*a*) and 1(*b*) for the first two cases. The points A, B and C to which the supply voltage is connected at the higher speed are the centre points of the phases at the lower speed. At this lower speed these points are therefore isolated, and it is possible to put extra windings in series for the higher speed only by joining them to the points A, B and C. These extra windings will be out of circuit when the machine is connected for the lower speed.

The supply voltage for the higher speed is connected to the three outer ends A′, B′ and C′ (see Fig. 2) of the extra windings,

Written contributions on papers published without being read at meetings are invited for consideration with a view to publication.
Prof. Rawcliffe is Professor of Electrical Engineering, University of Bristol.
Mr. Burbidge was formerly in the Electrical Engineering Department, University of Bristol, and is now with the English Electric Co., Ltd.

* British Patent Application No. 17216, 1957.

Table 1

BASIC 2 : 1 POLE-CHANGING CONNECTIONS
(With ideal auxiliary winding details)

Connection No.	Connection		Flux density in ⅄ connection, relative to other connection as unity		Auxiliary winding	
	p poles 60° spread	$2p$ poles 120° spread	p poles	$2p$ poles	Differential or cumulative	Increase of effective turns needed to equalize flux
1	⅄	Y	$\frac{\sqrt{3}}{\sqrt{2}} = 1{\cdot}22$	1·00	Cumulative	% + 22
2	⅄	Δ	$\frac{1}{\sqrt{2}} = 0{\cdot}707$	1·00	Differential	− 29
3	Y	⅄	1·00	$\frac{4\sqrt{2}}{\sqrt{3}} = 3{\cdot}26$	Cumulative	+ 226
4	Δ	⅄	1·00	$\frac{4\sqrt{2}}{3} = 1{\cdot}88$	Cumulative	+ 88

and it is still only necessary to have six external leads (A′, B′, C′, *a*, *b*, *c*) in order to control the machine. Since the cost of control gear is often comparable with the cost of the machine controlled, any improvement in a motor which involves no change whatever in the associated control gear is always particularly welcome. The principle of the modification proposed is to add extra winding turns in series with the parallel-star connection, in the way shown in Fig. 2, in order either to equalize the air-gap flux densities for the two pole numbers, or at least to make them less unequal. These turns can be connected to act either differentially or cumulatively, the former being the correct alternative when the flux density would otherwise be too small, and the latter when the flux density would be too great.

It is clearly desirable that as few extra winding turns as possible should be used, and therefore the modification should normally be based on that particular one of the four standard 2 : 1 pole-changing connections which itself gives the nearest approach to equality of air-gap flux densities at the two speeds. Table 1 shows the relative flux densities for the four possible connections, the first two of which are now seen to be clearly preferable to the other two in all ordinary circumstances, as stated above. Further, it is clear that the connection in parallel-star/series-star is the best from which to derive the proposed modification; and that the additional turns ought to be connected cumulatively, to reduce the relative flux density at the higher speed. As will be seen from Table 1, the ratio of the number of effective additional turns per pole to the effective normal turns per pole should ideally be 22%, based on the first connection.

(3.1) Design of New Winding

It has been shown in Table 1 that, to obtain equal flux densities in the air-gaps at the two speeds, the number of effective turns at the higher speed has to be increased in the ratio $\sqrt{3} : \sqrt{2}$. (This ignores the reluctance of the iron portion of the magnetic circuits.)

Suppose n = Number of conductors per slot in the basic winding, full-pitched for 2 p poles.
n_1 = Number of conductors per slot in the auxiliary winding.
Z = Total number of slots.

(Uniform spread throughout is assumed.)

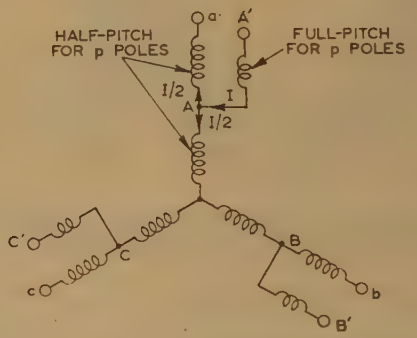


Fig. 2.—New 2 : 1 pole-changing connection.

p poles	Supply	A′ B′ C′	⊙ Six control leads
	Join	a b c	
$2p$ poles	Supply	a b c	
	Disconnect	A′ B′ C′	

Note that current per conductor in the auxiliary winding is double that in the main windings.

The number of effective conductors in series per phase for p poles, in the basic winding alone, connected in parallel-star is

$$\frac{[(\text{Conductors per phase}) \times (\text{Chording factor}) \times (\text{Spread factor})]}{(\text{Paths in parallel})} = \frac{nZ}{3} \times \frac{1}{\sqrt{2}} \times \frac{3}{\pi} \times \frac{1}{2}$$

The number of effective conductors per phase in the auxiliary winding, assumed to be full pitch for p poles, is

$$\frac{n_1 Z}{3} \times 1 \times \frac{3}{\pi}$$

Therefore

$$\frac{n_1 + \frac{n}{2\sqrt{2}}}{\frac{n}{2\sqrt{2}}} = \frac{\sqrt{3}}{\sqrt{2}}$$

i.e.

$$\frac{n_1}{n} = \frac{\sqrt{3} - \sqrt{2}}{4} = 0{\cdot}0795$$

This shows that the number of auxiliary turns required is only about 8% of the turns in the main winding. However, as the current per turn in the auxiliary winding is double that in the main winding, the slot space required is about 16% of the slot

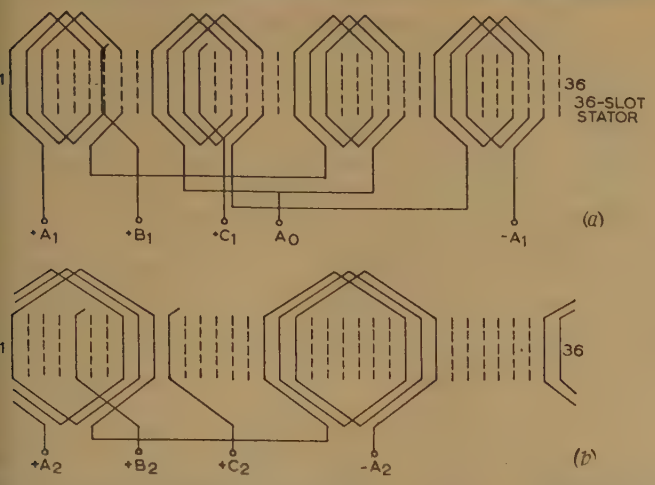


Fig. 3.—Winding diagram of one phase of new 8 : 4 pole-changing winding.

All phases are identical. The starting-points of other phases are shown.
(*a*) Main winding (two layer) full-pitched for 8 poles, each coil wound with 40 turns of No. 21 s.w.g. wire.
(*b*) Auxiliary winding (single layer) full-pitched for 4 poles, each coil wound with 5 turns of No. 18 s.w.g. wire.
A_0 and $+A_2$ are normally permanently connected.

space occupied by the main winding, i.e. about 14% of the whole slot space. The actual design of the winding of an experimental machine is given in Fig. 3.

The number of turns needed in the auxiliary winding when the machine has a small number of slots per pole per phase is somewhat different from the number needed where the winding can be assumed to be uniformly spread. As the auxiliary winding in effect is the difference term in an equation, the variation in the number of auxiliary turns is rather greater than might have been expected. However, starting from a calculated value, the number of turns would always be adjusted from test results to give nearly coincident magnetizing curves, as shown in Fig. 4, and the variation is not of great consequence.

As the flux density in the core is necessarily twice as large for the smaller number of poles as for the larger, the effect of the iron portion of the magnetic circuit will be greater in the former case. It may therefore happen that, in order to make the magnetizing curves for the two speeds coincide, it is necessary to use a few more turns in the auxiliary winding than simple theory would suggest.

If the use of coils of two different sizes were held, on constructional grounds, to be objectionable, it would be possible to use auxiliary coils of the same size as the rest of the winding, but the number of turns would have to be increased in the ratio $\sqrt{2}$, i.e., about 20% of the winding space would be taken up by the auxiliary winding. This would correspondingly worsen the continuous ratings of the machine, though the torque properties would, of course, be unaltered.

(3.2) Theoretical Ratings of the New Winding

The auxiliary winding can be full-pitched for the smaller number of poles, instead of being half-pitched, as is the rest of the winding; and the average chording factor for the higher speed is therefore given by the resultant of $(\sqrt{3}-\sqrt{2})n/4$ turns, full-pitched, in series with $n/2$ turns of chording factor $1/\sqrt{2}$.

Hence the average chording factor is

$$\left[\frac{\frac{\sqrt{3}-\sqrt{2}}{4}n+\frac{n}{2}\frac{1}{\sqrt{2}}}{\left(\frac{\sqrt{3}-\sqrt{2}}{4}+\frac{1}{2}\right)n}\right]=\frac{\sqrt{3}}{\sqrt{3}-\sqrt{2}+2}=0{\cdot}748$$

Since the flux density has its full value, the most pessimistic estimate of the power rating at the higher speed would be 75% of the full rating of the frame for that speed. This, however, assumes that the power rating for full flux is reduced in direct proportion to the stator chording factor, and thus assigns quite undue weight to the effect of stator copper loss. It is probable that 90% of the full frame rating is a better estimate, and this figure was actually exceeded on test. The continuous rating obtained in a particular case (see Section 4) was 4·95 h.p., which compared with 5·35 h.p. for the same frame wound normally for the smaller pole number, i.e. 92·5% of full rating was obtained.

At the lower speed, the auxiliary windings AA′, BB′ and CC′, which take about 14% of the winding space, are out of circuit; and the active copper section at this speed is therefore reduced by a proportionate amount, compared with the available section if the machine were wound for this speed alone. The permissible current loading for the same stator copper loss is thus reduced to approximately $\sqrt{0{\cdot}86}$ (= 93%) of what it would be with the full winding section. Since the flux density again has its full value, the power rating of the machine at the lower speed should be about 93% of what it would be if the machine were wound, with the same spread factor, for that speed alone. The instantaneous torque rating is unaffected by stator copper section and is equal to the full rating of the frame.

A 2 : 1 pole-changing motor, however, necessarily has a spread of 120° at the lower speed, which reduces its rating by about 10% compared with a machine wound with the usual spread of 60°. The rating of this new winding theoretically should therefore be about 84% of the full rating of the frame for the larger number of poles. The rating actually obtained in a particular case (see Section 4) was 1·55 h.p., which is 80% of the full frame rating when wound with a simple 60° spread winding for the higher pole number only.

There seems to be a widespread instinctive prejudice against any winding in which some part lies idle under certain conditions; but, in fact, it is much better to reduce the number of effective turns in a winding by omission rather than by chording. Any chorded winding gives a reduction in turns number without reducing the I^2R loss; the omission of turns reduces the heat generated as well. This winding, for example, effectively wastes 14% of the copper space at the lower speed, and yet this is less disadvantageous than chording to two-thirds of full pitch, which designers regard as a standard routine for machines of many kinds.

(4) TEST RESULTS

The effectiveness of the auxiliary winding should first be verified by taking magnetizing curves at the two speeds, and checking that they are nearly coincident. (It is also instructive to take a magnetizing curve for the smaller number of poles with the auxiliary winding omitted.) If the first two curves are not nearly coincident, the number of turns in the auxiliary winding needs adjustment. The greater the number of auxiliary turns, the further does the corresponding magnetizing curve rise, in the direction of the voltage axis.

Such tests were performed on an experimental machine with the results shown in Fig. 4.

In this machine, the number of turns initially chosen for the auxiliary winding proved to be nearly correct; and the machine was therefore subjected to comprehensive tests with the results given in the last row of Table 2. This Table also shows comparable figures for the same machine frame when wound with other orthodox types of single-speed and two-speed windings.

It will be seen that the performance of a machine wound with the new winding is better than that of a parallel-star/star

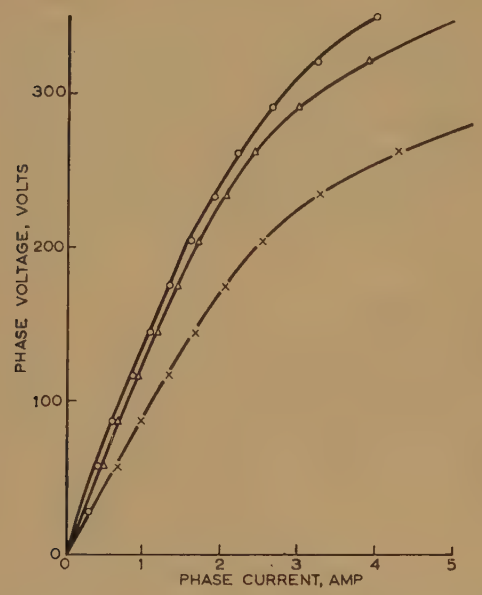


Fig. 4.—Magnetizing curves of new 8 : 4 pole-changing winding.
○—○—○ 8-pole operation.
△—△—△ 4-pole operation, using auxiliary winding.
×—×—× 4-pole operation, auxiliary winding omitted.

(5) OTHER POSSIBLE MODIFICATIONS

Perhaps the most popular of the four standard connections given in Table 1 has been the second. This is probably because there is a tendency to prefer optimum performance (especially in relation to torque) at the lower speed, at the cost of a less good performance at the higher speed, rather than the reverse. Of the three connections which give full flux at the lower speed, the second connection (Table 1) gives the highest flux at the higher speed. There is a further reason why it may even be advantageous to have a higher air-gap flux density at the lower speed. The flux density in the core, for a given air-gap flux density, is twice as great at the higher speed as it is at the lower. Full flux density in the air-gap at the higher speed, in principle, involves a bigger core and frame than would have been necessary had there been a reduced flux density at this speed.

If it is desired exactly to equalize the fluxes at the two speeds, it is plainly best to start with the first connection in Table 1, for which the fluxes are most nearly equal to start with. However, if exact equality is not desired, it may be sufficient to add a few differentially connected turns, in series with the double-star at the higher speed, in order to derive a modified form of the second connection in Table 1, giving some increase of flux at the higher speed. Alternatively a limited number of cumulatively connected turns may be added, in series with the double-star at the lower speed, to give a modified form of the fourth connection in Table 1, again making possible a higher flux at the higher speed.

Table 2

COMPARATIVE TEST RESULTS OF NEW WINDING AND STANDARD WINDINGS

Winding and connection details	Continuous rating		Pull-out torque / Full-load torque		Power factor		Efficiency	
	h.p.						%	
Wound normally for 4 poles only (60° spread)	5·35		2·05		0·87		84·8	
Wound normally for 8 poles only (60° spread)	1·95		1·75		0·66		70·5	
	Higher speed	Lower speed	Higher speed	Lower speed	Higher speed	Lower speed	Higher speed	Lower speed
Wound for ⅄⋊/⅄ pole changing 4/8 poles (60°/120° spread) voltage limited by 4-pole connection	4·50	1·35	2·20	1·45	0·86	0·70	76·1	66·1
Wound for ⅄⋊/Δ pole changing 4/8 poles (60°/120° spread) voltage limited by 8-pole connection	3·15	1·80	1·50	1·80	0·89	0·66	75·4	69·7
New winding described in the paper ..	4·95	1·55	2·10	1·90	0·86	0·65	81·2	67·7

connected machine at the higher speed, and it is almost as good as the performance of a parallel-star/delta connected machine at the lower speed—indeed, in respect of pull-out torque it is better. In short, the new winding combines the advantages of both connections and the disadvantages of neither.

The ratings of the frame for the two speeds even compare not unfavourably with the ratings of the same frame when wound normally for one speed only; first for the higher speed and then for the lower. In theory, the new winding ought to give a continuous output of about 4·80 h.p. at the higher speed as compared with 5·35 h.p. for the same frame wound only for that speed. For the lower speed, the corresponding continuous ratings are 1·65 and 1·95 h.p. In practice, the test ratings at the two speeds were, respectively, 4·95 and 1·55 h.p., as stated in Section 3.2.

The use of differentially connected turns is wasteful and would only be considered if a moderate increase in flux at the higher speed would be sufficient. The use of the delta/parallel-star connection (No. 4 in Table 1) is not very common, but if it were desired to employ it, the two flux densities could be made a little less unequal by cumulatively connected turns in series with the parallel-star connection at the lower speed. These turns would, however, have to be full-pitched for the larger number of poles, which is different from the main modification discussed in the paper.

(6) ACKNOWLEDGMENT

The authors are indebted to Mr. A. Walsh, Consulting Engineer—Metropolitan-Vickers Electrical Co., Ltd., for helpful discussion in the course of the work.

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The Institution of Electrical Engineers
Paper No. 2433 U
Dec. 1957
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LINEAR INDUCTION MOTORS

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(*The paper was first received* 21*st May, and in revised form* 12*th August*, 1957.)

SUMMARY

The first half of the paper is concerned with the fundamental differences between linear induction motors and their more conventional rotating counterparts. In particular, the effects of the discontinuities which are necessarily introduced either in the stator ('short stator' effects) or in the rotor ('short rotor' effects) are discussed theoretically. Such discontinuities are shown to involve transient conditions in a linear motor, even when the machine runs at constant speed. The exploitation of double-sided and tubular stators is also discussed.

The second part of the paper deals with practical applications of linear induction motors. It includes a brief account of the development of a linear motor which produces a reciprocating motion of stable amplitude without the necessity for any switching device. The possible applications of this type of motor to the textile industry are examined. The use of linear motors as liquid-metal pumps is examined in the light of the earlier theory, and the final section suggests an unusual outlook on induction-motor design which was inspired by the study of linear motors.

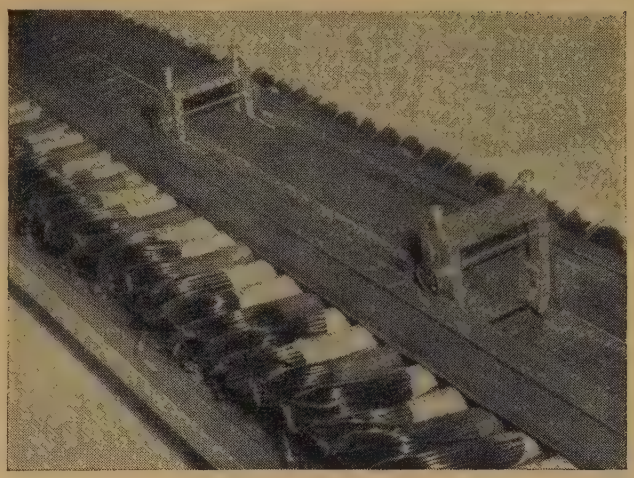


Fig. 1.—A linear induction motor.

(1) INTRODUCTION

The electrical engineer is usually taught the theory of windings in electrical machines by means of 'developed' diagrams. In a developed diagram, one imagines the cylindrical machine to have been cut by a radial plane and opened out flat. The diagrams generally give a view of the inside of the stator. The usefulness of a developed diagram lies in the fact that the entire winding can be clearly drawn on a flat sheet of paper. An example of a more complicated form of developed diagram is a map projection, where the object is to present a flat reproduction of shapes on a spherical surface.

There is no reason why the picture of a developed stator cannot become a reality, simply by inserting a set of windings into slots in blocks of straight laminations, as shown in Fig. 1. Any type of rotary electrical machine can be manufactured as a developed or 'linear' machine. In the case of motors, the torque and angular acceleration of the conventional rotary machine are replaced by force and acceleration in a straight line. It should be remembered, however, that all the other parts of the motor, apart from the stator winding, must also be reproduced in linear form. A linear Schrage motor, for example, would require a linear commutator with brush pairs, together with three slip 'tracks' corresponding to the slip rings of the rotary version.

Discussion of new techniques invariably results in the coining of new words and phrases. The word 'linear' has already been used in connection with particle accelerators, but as there is little likelihood of confusion between these devices and electric motors, there appears to be no objection to the use of the word for the latter. It has been suggested that the word 'rotor' is quite unsuitable for the moving portions of linear machines. The word 'linor' has been suggested as alternative. Familiarity with new words takes time to acquire and since there is no ambiguity introduced by retaining the word 'rotor', even for linear machines, this procedure has been adopted throughout the paper.

Just as a very large percentage of the total horse-power of all rotary electric motors which have ever been manufactured is made up of induction motors, so, in the case of linear machines, the induction motor is the one which has attracted most attention. The main reason for this is undoubtedly that there is no electrical connection to the moving part in the simplest form of induction motor. The present paper deals only with induction motors, although some of the arguments on the design of such motors can readily be applied to other types.

(2) FUNDAMENTAL PRINCIPLES OF THE LINEAR INDUCTION MOTOR

(2.1) The Travelling Magnetic Field

As with rotary machines, the ampere-turns of the primary winding can be arranged to give a close approximation to a sinusoidally distributed flux in a uniform air-gap. For a 3-phase stator, the flux density b at a distance x from a suitably chosen point on the linear array at time t may be written as

$$b = \frac{3}{2} B_{max} \cos\left(\omega t - \frac{\pi x}{s}\right)$$

Where B_{max} is the maximum flux density capable of being produced by each phase, ω is the angular frequency of the supply, and s is the pole pitch. This is a travelling wave of flux density whose phase velocity, v_s, is $\omega s/\pi$, which, writing $\omega = 2\pi f$, yields

$$v_s = 2sf \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (1)$$

In the case of conventional rotary machines the number of pole pitches around the periphery is always a whole even number, and it is usual to denote the number of poles in a machine by $2p$.

Written contributions on papers published without being read at meetings are invited for consideration with a view to publication.

This is an 'integrating' paper. Members are invited to submit papers in this category, giving the full perspective of the developments leading to the present practice in a particular part of one of the branches of electrical science.

Dr. Laithwaite is in the Electrical Engineering Department, University of Manchester.

Hence the angular velocity n_s in revolutions per second is given by

$$n_s = \frac{f}{p} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (2)$$

Confusion between this equation and eqn. (1) should be carefully avoided.

(2.2) Discontinuities in Linear Motors

To say that the only difference between a rotary machine and its linear counterpart is that the linear version has a start and a finish, whereas the rotary machine is a continuous loop, appears to be merely stating the obvious, but the statement is more profound than it might at first appear.

Linear induction motors may be divided into two classes, namely

(*a*) Those in which the essential discontinuities occur at the ends of the stator and which are termed 'short-stator' machines.
(*b*) Those in which the discontinuities occur at the ends of the rotor and which are termed 'short-rotor' machines.

The difference between (*a*) and (*b*) is perhaps best emphasized by a diagram. Fig. 2(*a*) shows a short-stator machine, and Fig. 2(*b*)

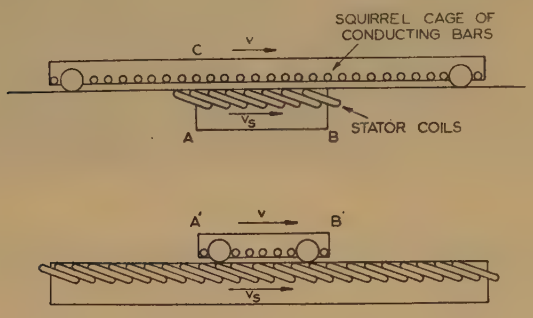


Fig. 2.—Types of linear motor.
(*a*) Short stator.
(*b*) Short rotor.

a short-rotor machine. It is assumed that, in the normal operation of the machine shown in Fig. 2(*a*), the stator block AB is always entirely covered by rotor bars, whilst the machine shown in Fig. 2(*b*) is always operated so that the whole of the rotor A′B′ is contained within the stator track. It should be also remembered that the functions of rotor and stator can be interchanged, as in rotary machines, and that the portions of the machines shown in Fig. 2 which are merely squirrel cages may, in fact, be the stationary parts.

Notwithstanding, such machines will be referred to as 'short-stator' or 'short-rotor' machines in accordance with the convention of Fig. 2.

The basic disadvantage of the linear motor is apparent from Fig. 2. Either a long squirrel cage of conducting bars must be provided, of which only a short section is in use at any one time, or a long stator winding must be provided merely to drive a short rotor. In the latter case, if switching is to be avoided, the whole of the track must be energized, even though only a small section is covered by rotor bars. Despite these obvious extravagances, the use of a linear motor has been considered justifiable in certain cases. Two examples of very large linear motor projects are the smokeless gun, which was developed during the First World War using a short-rotor machine,[1] and the Electropult for providing assisted take-off for aircraft.[2] In the latter example the runway was over a thousand feet in length, whilst the moving carriage was only of the order of 10 ft in length. The linear motor, therefore, had to be of the inverted short-stator type, in which the short carriage carrying the aircraft was fitted with a set of coils fed via 'slip tracks' from a 3-phase supply.

The two types of linear motor represented in Fig. 2 are basically different, and the theory of each will therefore be considered separately.

(2.3) Short-Stator Machines

In the first instance the action of the machine shown in Fig. 2(*a*) will be considered when the rotor speed, v, is equal to the synchronous speed v_s, calculated in accordance with eqn. (1). At the instant shown in Fig. 2(*a*), all the rotor iron to the left of the point C contains no flux. Each rotor tooth is contained within a short-circuited loop of conductor and contains no flux even as it is about to enter the stator block AB. Provided that the teeth are very small and the edge A of the stator provides a sharp discontinuity, each tooth effectively enters the stator area in an infinitely short time. There can therefore be no change of flux through the conducting loop without an infinite voltage. However, if the stator is fed with a constant current, then, as each particular tooth enters the stator block, an equal and opposite current flows in the rotor conductors surrounding the tooth. As the rotor conductors have resistance, a rate of change of flux is necessary to maintain the resistive voltage drop round the loop, and, for a current-fed stator, the flux in the rotor tooth builds up at a uniform rate as the tooth progresses from A to B. At B a reversal of the procedure is effected, the tooth containing the flux being unable to change its flux instantaneously, and the rotor current reverses in order to maintain the flux immediately outside the block. This current then decays according to the rotor time-constant outside the block. Both edges of the stator are thus instrumental in producing losses in the rotor. The rotor current under the block results in rotor copper loss, whilst producing no useful torque. The exit edge loss may be considered to be either magnetic energy carried away at rotor velocity or rotor copper loss occurring outside the block. The result of this loss of energy at the exit edge is a back torque which, in practice, prevents the rotor from ever attaining the calculated synchronous speed.

When the rotor speed is no longer synchronous the conditions are more complicated. A recent paper contains an analysis[3] in which the flux distribution under a stator block is calculated in terms of the slip, assuming the stator to be fed with a constant current.

Such an analysis is relevant to a series-connected machine in which the same current flows through all coils on the stator block of the same phase. Furthermore this statement is true even if the machine is fed from a constant-voltage source, so long as all the coils of each phase are in series. Considerable confusion has arisen in the past over this point in connection with the analysis of Reference 3, and some further explanation is perhaps useful.

The theory is based on the fact that all the stator coils produce a travelling wave of current density of constant amplitude J_s without specifying the method of connection. For a series-connected machine this implies that each phase carries a constant current, say I_s. On this basis, such characteristics as the speed/torque curve can be calculated. Knowing the method of connection, another characteristic which can be predicted is that of the terminal-voltage/slip curve. This last curve enables prediction of the machine operation at constant voltage to be made, as in the following example for the speed/torque characteristic:

Torque at slip σ_1 on constant current $I_s = T_1$
Torque at slip σ_2 on constant current $I_s = T_2$
Terminal voltage at slip σ_1 on constant current $I_s = V_1$

Terminal voltage at slip σ_2 on constant current $I_s = V_2$

Current at slip σ_1 when the terminal voltage is $V = \frac{V}{V_1} I_s$

Current at slip σ_2 when the terminal voltage is $V = \frac{V}{V_2} I_s$

Torque at slip σ_1 when the terminal voltage is $V = \left(\frac{V}{V_1}\right)^2 T_1$

Torque at slip σ_2 when the terminal voltage is $V = \left(\frac{V}{V_2}\right)^2 T_2$

Such computations have been performed for series-connected short-stator machines in Reference 3.

If the coils of a short-stator machine are all identical and parallel-fed from a constant-voltage source, the parallel connection attempts to force the flux to a finite value at the edge of the stator block where the rotor conductors are entering. This results in a very large current in the end coils, which invariably over-heat, and parallel-connected stators require a special design of end coils. This technique has become known as 'end grading'.

Possibly the simplest way to regard the action of the short-stator machine, when running at speed $v \neq v_s$ and series-connected, is to consider the effect of rotor and stator currents separately. At the entry edge, a current equal and opposite to the stator current is induced in the rotor. Whilst the travelling wave of stator m.m.f. moves across the block at synchronous speed, the rotor current, set up transiently at the entry edge, moves at rotor speed and therefore moves alternately in and out of phase. The more rigorous theoretical approach of Reference 3 yields the flux distributions shown in Fig. 3 for an n-pole block at various slips. The phase of the flux has been indicated by resolving the flux density into components B_p and B_q in phase with, and in quadrature with, the stator current. Conventional magnetizing current and leakage are neglected in the theory responsible for these curves. It follows that, with these assumptions, rotor current under the stator block is also constant, as are rotor and stator copper losses.

A useful comparison can be made between a short-stator machine and an equivalent conventional machine. The equivalent conventional machine is defined in Fig. 4. The dotted lines in Fig. 3 indicate the component of the flux density B_p in the equivalent conventional machine. The power output is given by

$$\frac{1}{2}\int_0^{ns} B_p J_s dS$$

where J_s is the constant stator current density and S is the distance from the entry edge of the block and hence is proportional to the area under the B_p curve. For a slip $\sigma = 1/(2n + 1)$ [Fig. 3(*b*)], the shaded area under the B_p curve is less than that contained in the rectangle ABCD under the dotted curve, This means that, for fixed losses, the output of the linear machine is less than that of the conventional machine, and the efficiency is therefore lower. The extra losses have been termed 'excess rotor copper loss'. The area under the B_q curve is a measure of the reactive power. For a slip $\sigma = 1/(n + 1)$ [Fig. 3(*c*)], the loss of output over the first half of the block is completely made up by the second half and there are no excess rotor copper losses. The power factor of the linear machine is not unity, however. For a slip between $1/(n + 1)$ and $2/(n + 2)$ [Fig. 3(*d*)], the excess rotor copper loss is negative and the linear machine, so far as excess rotor copper loss is concerned, is better than the equivalent conventional motor. The power factor is higher than at slip $1/(n + 1)$ but still less than unity. For a slip

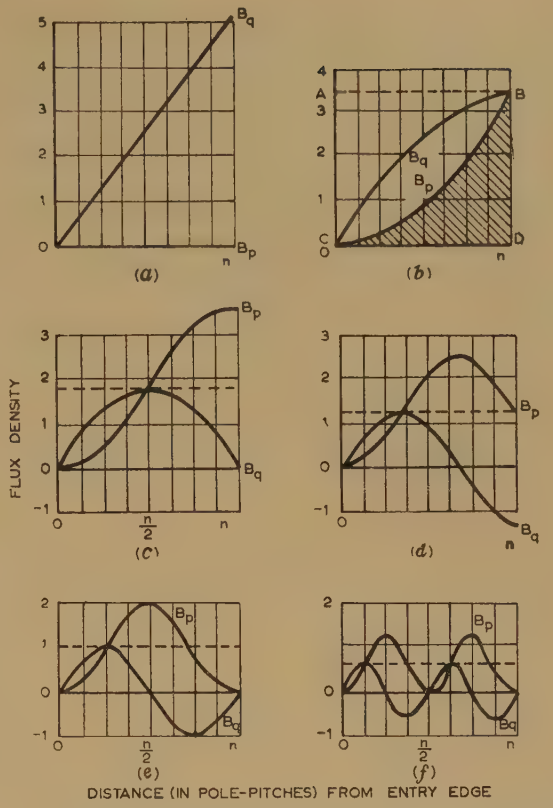


Fig. 3.—Flux distributions in a short-stator machine at various slips.

(*a*) $\sigma = 0$ (*d*) $\sigma = \frac{3}{2n + 3}$

(*b*) $\sigma = \frac{1}{2n + 1}$ (*e*) $\sigma = \frac{2}{n + 2}$

(*c*) $\sigma = \frac{1}{n + 1}$ (*f*) $\sigma = \frac{4}{n + 4}$

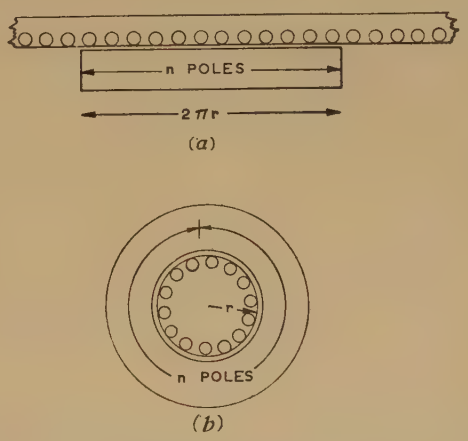


Fig. 4.—Basis of comparison between linear and rotary machines.
(*a*) Short-stator machine.
(*b*) Equivalent conventional machine.

$2/(n + 2)$ [Fig. 3(*e*)], there is no flux at either entry or exit, no excess rotor loss and unity power factor.

In assessing the relative merits of the machine in this way the exit edge loss has been ignored. This loss is proportional to the square of the flux density at the exit edge and to the rotor velocity. It is therefore a maximum for $\sigma = 0$, and does not fall to zero until the slip has increased to $\sigma = 2/(n + 2)$, when there is no

flux at the exit edge, as shown in Fig. 3(*e*). The exit edge loss at any slip can be reduced by including some unexcited iron beyond the end of the stator winding.

(2.3.1) New Concepts introduced by the Short-Stator Machine.

It is clear from the foregoing theory that short-stator machines cannot be designed on the basis of conventional machines. The flux density under the block is non-uniform and its distribution varies with slip. Rotor copper loss cannot be calculated by the simple formula (slip × torque × synchronous speed). Power factor may be less than unity even though magnetizing current and leakage reactances have been neglected. This last fact sets a problem as to the mechanism of the stored energy. The explanation of this and other unusual phenomena may be found in the one statement that every part of a short-stator machine is working under transient conditions at all times. The normal equivalent circuit of an induction motor cannot therefore be used. Magnetic field is continually being set up along the block, carried away and its energy subsequently dissipated outside the block.

The most interesting conditions obtaining in short-stator motors are undoubtedly those between $\sigma = 1/(n+1)$ and $2/(n+2)$, including the latter condition. The linear motor shown in Fig. 4 is indistinguishable externally from the equivalent conventional machine for $\sigma = 2/(n+2)$, despite its inherent transient behaviour and non-uniform flux distribution. Between the slips just quoted, excess rotor copper loss is negative and may be of such a value as to more than compensate for the extra exit edge loss introduced by a non-zero exit flux density. The engineer is accustomed to calculating the effects of transient phenomena but, generally, he does it with the attitude that such phenomena are a necessary evil. The short-stator machine is just one example of a machine transiently operating at all times. It has been shown that this type of motor, under particular conditions of running [$\sigma = 2/(n+2)$], can be made as good as a conventional motor of the same output and pole pitch. The study of the short-stator motor with its inherent transient behaviour has possibly opened the way to a new line in machine development. Only by using short-stator machines can the new principle of brushless variable speed[3] be exploited.

The short-stator principle is being applied with advantage in Russia, not to linear motors but to rotary machines whose stators cover only a fraction of the surface of the rotor. The arc subtended at the centre is normally less than 100° and the machines are referred to as 'arc' or 'arch motors', or as 'motors with open magnetic circuit'.[4,5,6,7,8] All attempts to produce a brushless induction motor capable of speeds in excess of 3000 r.p.m. when fed from 50 c/s mains have so far been unsuccessful. If such a machine is ever to be produced it seems highly probable that it will fall into the transiently operating class. What other advantages may be gained from the manufacture of machines deliberately arranged to be transiently operating cannot yet be foreseen.

Some fairly obvious design criteria emerge from the foregoing discussion on short-stator motors, series-connected. For a reasonably high efficiency, the excess rotor loss must be cut to a minimum and a full-load slip of at least $1/(n+1)$ is desirable. At this slip, excess loss is zero and rotor copper loss can be calculated from the simple formula, yielding rotor efficiency of $(1-\sigma)$. Thus rotor efficiency is $[1 - 1/(n+1)] = n/(n+1)$ and it is clear that n should be as large as possible. One important factor has been neglected in the simple theory, however. If magnetizing current is taken into account, the flux curves shown in Fig. 3 are modified. For example, at a large slip the flux distribution takes the form shown in Fig. 5, where B_p and B_q degenerate with rotor time-constant towards their respective

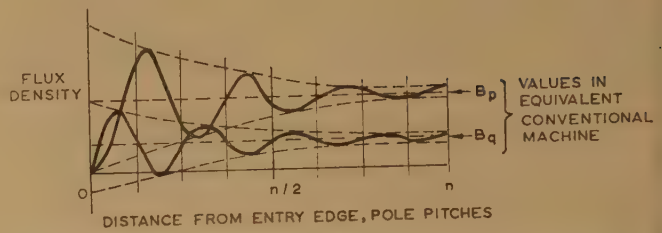


Fig. 5.—Flux distributions in a short-stator machine with finite magnetizing current.

values in the equivalent conventional machine. The flux distribution in very long linear machines therefore tends to degenerate to the conventional machine pattern, yielding characteristics which conform more closely to those predicted by the equivalent circuit.

(2.4) Short-Rotor Machines

In the short-rotor machine the overall performance depends very much on the inactive coils outside the rotor, since there may easily be more of these than there are active coils. Fig. 6 shows

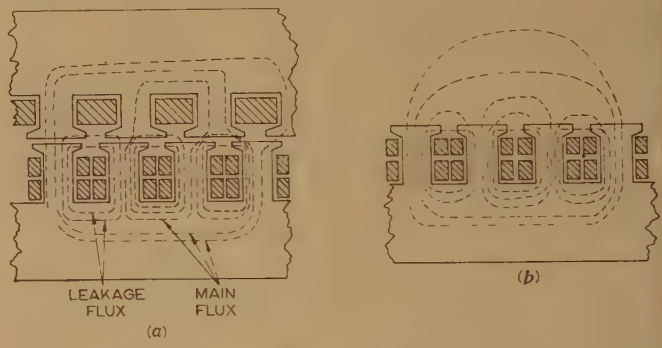


Fig. 6.—Flux paths in a short-rotor machine.
(*a*) Active section of track.
(*b*) Inactive section of track.

the flux paths when the rotor is present and when it is removed. The only paths for the flux from an open-sided stator are the normal stator leakage paths and very long paths through the air. In the absence of saturation the inductance of the coils outside the rotor is about the same as the leakage inductance of the active coils. If the rotor occupies a length $1/\alpha$ times the length of the stator and the latter is series-connected, the whole machine may be represented as a conventional motor whose stator impedance is $R_s + jX_s$ in series with a resistance αR_s and reactance αX_s, neglecting transient effects. If the stator is parallel-connected the conventional motor is in parallel with an impedance

$$\frac{R_s}{\alpha} + \frac{jX_s}{\alpha}$$

Neither arrangement is satisfactory from an efficiency or power-factor point of view. The stator current in the inactive coils is of the same order as that in the active coils and the stator copper losses of the active part of the machine are multiplied by α. The reactive power due to the leakage reactance of the active part is also effectively multiplied by α when considering the remainder of the stator winding.

In short-stator machines, the transient phenomena occurring at the edges are all-important in setting the pattern of behaviour. In the case of short-rotor machines, some transient phenomena

must, in practice, occur at the rotor edges, but the effect is believed to be secondary compared with that in short-stator machines, and no theoretical work has so far been attempted on this aspect.

If the rotor of a short-rotor machine is called upon to leave the end of one stator track and enter another, transient phenomena of the short-stator type will, of course, occur as the rotor crosses the stator discontinuities.

(2.5) Double-Sided Stators

The mechanical difficulties of feeding the rotor of a conventional rotating induction motor both from inside and outside prevent the advantages of such a system from being exploited. In linear machines, however, double-sided drive is a natural choice since the resulting symmetry produces a number of advantages which are in no way offset by difficulties of construction. If the rotor contains iron, for example, a single-sided system such as either of those shown in Fig. 2 produces a very large magnetic force on the rotor perpendicular to the track, adding considerably to the mechanical difficulties in the design of wheels and track rails. The magnetic force in C.G.S. units is $B^2/8\pi$ dynes/cm^2, whereas the driving force is $BJ/10$ dynes/cm^2, where J amperes per centimetre is the current density. For all but very large machines, the magnetic force is a whole order of magnitude bigger than the driving force. A double-sided drive, as shown in Fig. 7, reduces the magnetic force to something much



Fig. 7.—Arrangement of a double-sided linear motor.

less than the driving force, provided that the rotor is maintained in a reasonably symmetrical position between the upper and lower stator tracks. Apart from reducing magnetic attraction, there is little advantage in a double-sided track in which corresponding poles on upper and lower halves are both of the same instantaneous polarity. The main advantage of the double-sided track lies in the ability to obtain twice the m.m.f. per pole using the same size of rotor, by connecting opposite poles in upper and lower halves so that they assist each other in driving flux through the rotor. Since the magnetic circuit in such a system is complete, even without the presence of a rotor, the system can be used in conjunction with non-ferrous rotors consisting simply of slabs or sheets of conducting material. The rotary counterpart of this type of linear motor is the drag-cup motor, although the latter is fed from one side only.

Most of the factors for assessing the quality of a motor, such as efficiency and power factor, can be judged by evaluating a quantity $\tau\omega$, where τ is the coupled time-constant of the rotor given by[3]

$$\tau = \frac{4s^2\mu_0}{\pi\rho_r g} \quad . \quad . \quad . \quad . \quad . \quad . \quad (3)$$

where ρ_r is the rotor resistivity, g is the air-gap length, and μ_0 is the air-gap permeability.

The machine designer is more accustomed to the use of an equivalent circuit from which he can derive quantities such as the magnetizing current and the locked rotor current. If the magnetizing inductance in the equivalent circuit is denoted by L and the rotor resistance by R the rotor coupled time-constant may be written simply as L/R, making $\tau\omega = L\omega/R$. Eqn. (3) is useful because it enables τ to be calculated in terms of actual machine dimensions. In Appendix 1 of Reference 3 an expression is derived for the flux density in a conventional motor whose stator current density J_s is constant. From this an equation for the torque T at any slip can be derived in terms of τ. This expression, which does not include the effects of leakage reactance, is as follows:

$$T = \frac{\frac{1}{2}\rho_r J_s^2}{v_s}\left[\frac{\sigma}{\sigma^2 + \left(\frac{1}{\tau\omega}\right)^2}\right] \times \text{Rotor surface area} \quad . \quad (4)$$

Torque/slip curves plotted from eqn. (4) are shown in Fig. 8. Since these refer to constant-current supply the copper losses

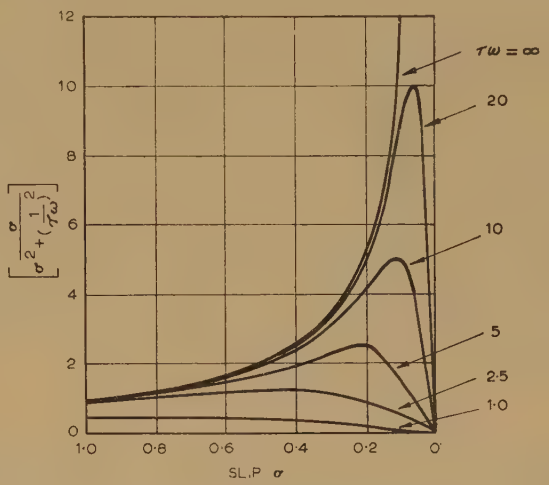


Fig. 8.—Effect of $\tau\omega$ on the speed/torque characteristics of an induction motor operating at constant current.

are constant, and if iron loss is ignored the peak efficiency occurs at the peak output. The higher this peak output can be made, the higher the efficiency. The significance of $\tau\omega$ in the calculation of efficiency is evident from Fig. 8. The effect of the magnetizing reactance on the power factor can be similarly derived. The relationship between no-load and locked rotor currents, and $\tau\omega$ is fairly simple. On no-load the stator resistance and leakage reactance may be considered small, compared with the magnetizing reactance, and the no-load current I_0 for a supply voltage V is

$$I_0 = \frac{V}{L\omega}$$

For the locked-rotor condition the magnetizing current may be ignored, and the locked-rotor current, I_s, is given by

$$I_s = \frac{V}{R + R_s}$$

where R_s is the stator resistance, and leakage reactances are ignored.

Thus

$$\frac{I_s}{I_0} = \frac{L\omega}{R + R_s}$$

If the stator resistance is taken to be equal to the equivalent rotor resistance, then to a first approximation $I_s/I_0 = \tau\omega/2$. For a reasonably good machine $\tau\omega$ would be greater than some constant of the order of ten. Applying this criterion to the non-ferrous rotor, ρ_r is inversely proportional to the thickness of the rotor, so that, provided that the air-gap is completely filled with conductor, $\rho_r g$ is constant for a particular conducting material. μ_0 is also constant making $\tau \propto s^2$. The resistivity of

copper being the lowest resistivity obtainable at reasonable cost, the minimum absolute pole pitch is virtually fixed for machines of this type which are to have a value of τ greater than a certain amount. In practice, the air-gap cannot be completely filled with conducting material, since mechanical clearance must be allowed, and the resistance of the rotor end bars must be taken into account in assessing ρ_r. Since end-bar length is proportional to s, an approximate formula for τ, taking into account mechanical clearance and end-bar resistance, is

$$\tau = \frac{4s^2\mu_0}{\pi k_g \rho_r \left(\frac{w+s}{w}\right) g}$$

where k_g is the ratio of distance apart of the tracks to rotor-conductor thickness, and w is the track width. To give some idea of the relative magnitudes of these quantities it is assumed that k_g can be made as low as 1·5 and that $w = s$. Then for 50 c/s supply $s > 12 \cdot 7$ cm yields a value of $\tau\omega$ greater than 10.

Since $\rho_r g$ is virtually constant, except for very small values of g, where mechanical clearance utilizes the major part of the gap, $\tau\omega$ is unaffected by choice of gap, once the pole pitch is decided. Choice of a very small gap results in a good magnetic circuit in the inactive parts of the track, so that parallel connection can be carried out without introducing a large magnetizing current from the inactive stator coils. This line of approach appears to be the only satisfactory one for the short-rotor machine.

(2.6) Tubular Motors

A natural extension of the double-sided linear motor is obtained by building a second pair of stators to operate along the sides of the rotor, thus enclosing the latter in a rectangular tube. Similar machines can be made with any number of tracks forming a tube whose section is a polygon and the inner faces of the iron can be curved to produce a circular-section tube, as shown in Fig. 9. This form of construction is generally preferable for a

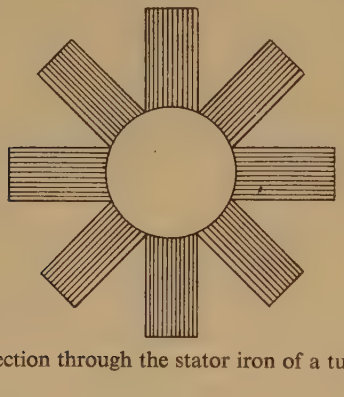


Fig. 9.—Section through the stator iron of a tubular motor.

tubular motor than any attempt to make each lamination of the stator iron lie in a radial plane. The big advantage of tubular motors lies in the saving in end turns. The flat stator of the linear machine, having been effectively rolled up about a longitudinal axis needs no end turns whatsoever and the stator winding is of the simplest form. One phase of such a winding is shown in Fig. 10. The rotors of tubular motors are more

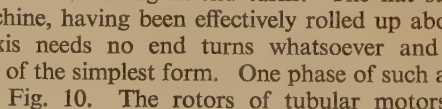


Fig. 10.—Winding arrangement for one phase of a tubular motor.

difficult to make, since ideally the iron laminations should each be wedge-shaped.

(2.7) Velocity Limitations

A large number of possible industrial applications of linear motors demand speeds in the range 1–10 ft/sec. These are very slow compared with the surface speeds of rotors of conventional rotating motors. For example, a 4-pole motor with a rotor diameter of 8 in has a surface speed approaching 50 ft/sec. It is well known that, in general, the smaller the diameter of a rotary induction motor the lower its efficiency and power factor for a given number of poles. A 20-pole machine with 8 in-diameter rotor (10 ft/sec peripheral speed) would be a poor machine. In linear motors, the pole pitch is likewise the limiting factor. The criterion $\tau\omega > 10$ may be applied to rotors containing iron as well as non-ferrous rotors, the only virtue in the iron being that the combination of low values of both ρ_r and g can thereby be obtained. There is a lower limit to ρ_r, however, for a given rotor thickness, even with iron.

The only method of improvement lies in a reduction of supply frequency. Eqn. (1) shows that $v_s \propto sf$ whilst $\tau\omega \propto s^2 f$, and hence for a given velocity v_s, $\tau\omega \propto s$, yielding a linear increase with pole pitch. The disadvantage of requiring a frequency changer in addition to the linear motor usually outweighs any other advantage the linear device may have.

(3) APPLICATIONS OF LINEAR INDUCTION MOTORS

(3.1) Shuttle Propulsion

One of the earliest suggestions for the application of linear induction motors occurred only seven years after the invention of the rotary induction machine.[9] The suggested application was in connection with weaving looms. It was proposed to use a linear track at each side of the loom to propel the shuttle, which was to be the rotor, from side to side. Reversed motion could be obtained by reversal of two phases of the supply. Such an application essentially involves the use of a short-rotor machine. Between that time and 1951 many patents involving the use of linear induction motors for shuttle propulsion were taken out. None of the schemes proposed proved to be very successful. Some involved single-sided tracks with all the disadvantages of magnetic attraction, inefficiency and low power factor, and these were obviously doomed to failure. All the systems suggested necessitated the use of switches for reversing the direction of motion. It is thought that this, more than any other single factor, prevented any of the schemes from succeeding, since the number of reversals required was of the order of 200 per minute and the cost of maintenance of the switching devices must have been a real problem to the inventors. Nevertheless the demands of a shuttle-propelling system are largely met by a linear induction motor, and in many ways the requirements of the shuttle are ideally suited to this type of motor. For example, the speed required is of the order of 50 ft/sec. The shuttle must be capable of free passage between the warp threads without any mechanical or electrical contact. These requirements can be met in part by means of a short-rotor type of motor, particularly if the rotor contains no iron. The shuttle, during the entire length of its travel across the loom, can be subjected to forces, either accelerating or decelerating, rather than being simply hit and left to its own devices, as it is with mechanical systems. Reduction of wear, maintenance and noise are all possible rewards for the development of the electrically-propelled shuttle. Considerations such as these led the author to develop a linear motor, capable of producing oscillatory motion without the use of any switching device.[10,11] Since then it has become apparent

that there may be other applications for a motor of this type. A brief account of the principle of the self-oscillating motor is given before discussing its possible applications.

(3.2) The Self-Oscillating Motor

Two stator tracks are arranged as shown in Fig. 11(*a*), producing magnetic fields travelling towards the centre at velocity v_s.

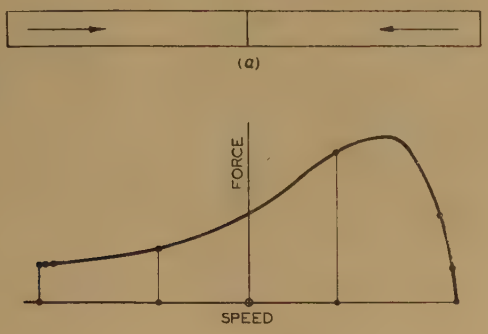


Fig. 11.—'Back-to-back' arrangement for self-oscillation.
(*a*) Schematic plan of tracks.
(*b*) Speed/force characteristic of the motor.

The speed/force characteristic of the motor is designed to be of the form shown in Fig. 11(*b*). A short rotor starting from rest at A is accelerated towards the centre by forces represented by the ordinates from 0 to *a*. At the centre a speed, *v*, is reached and the rotor enters the reversed field. In this field the initial rotor velocity is effectively $-v$, and the rotor is brought to rest by the action of forces represented by ordinates along *b*0. Since these forces are everywhere less than the accelerating forces, the point B where the rotor comes to rest is further from the centre than was A. On the next incursion to the centre a speed in excess of *v* is reached, and this build-up process continues until the rotor is effectively reaching synchronous speed as it crosses the centre. In this condition the amplitude of oscillation is stable.

One method of ensuring a rising speed/force characteristic, which is essential to the operation, is to exploit rotor leakage, but this is wasteful. A better method is to series connect the stator and produce a virtually constant-current device. In this case the peak force will occur at a slip $\sigma = 1/\tau\omega$. Thus, if $\tau\omega > 1$, stable oscillation is theoretically possible, though in practice $\tau\omega > 3$ gives a much more reliable type of motion. Expressions for the stable amplitude and period have been calculated and verified experimentally.[10] Thus, if f_m is the peak acceleration,

$$\text{Stable amplitude (no space between tracks)} = \frac{5}{6}\frac{v_s^2}{f_m}\tau\omega$$

$$\text{Time of one acceleration and deceleration} = \frac{4}{3}\frac{v_s}{f_m}\tau\omega$$

$$\text{Average speed} = \frac{5}{8}v_s$$

If the tracks are separated, the short-stator effect tends to assist in producing the required characteristic. Whilst the rotor is passing from one-half of the track to the other it is open-sided and rapidly loses its flux. Entry into the reverse field initially results in a very small operating flux and therefore virtually no retardation. Not until the rotor has progressed well into the track has a substantial retarding flux been built up. Having built up a flux, this is available for acceleration throughout the entire return journey. Provided that the rotor inertia is high, exit-edge back forces have little effect at the leaving edge.

Very often, applications requiring a reciprocating motion require a light rotor and low power input.

The use of thin, non-ferrous rotors in such applications is therefore important, both from the point of view of lightness and reduction of magnetizing current in the inactive portions of the track, as discussed in Section 2.5. The fact that the self-oscillating motion is wholly dependent on $\tau\omega$ exceeding unity suggests that better machines could be made using the technique of reducing frequency and increasing pole pitch described in Section 2.7. Two factors limit the extent to which this technique can be applied:

(*a*) For a fixed length of rotor the ratio of rotor length to pole pitch is decreased as pole pitch is increased. A rotor of length s/n, where n is greater than unity, experiences a force less than F/n, where F is the force which would be produced on a rotor of length s. The force per unit length of rotor decreases rapidly as n increases for constant flux density and rotor resistivity.

(*b*) For a fixed amplitude of oscillation the greater the pole pitch the fewer pole pitches are traversed on either side of the centre. When the semi-amplitude becomes less than one pole pitch, the rotor can hardly be said to be operating normally in a travelling field of speed v_s, particularly as the rotor is likely to 'carry over' flux from one field into the reverse field. Viewed in another way, when the frequency of the supply has been reduced to a value comparable with the frequency of the oscillation, some curious 'beat' effect is likely to occur. Modulation of the amplitude of oscillation has been observed experimentally under such conditions. It has also been established experimentally that a short-rotor, one third of a pole pitch in length, is capable of attaining normal running-light speed on a linear track, so that any irregularity of oscillation is thought to be due to 'carry-over' effects.

The question whether a self-oscillating motor with non-ferrous rotor can be applied successfully to any particular problem is therefore a matter of the demanded amplitude and speed only.

(3.2.1) Conservation of Rotor Energy.

Assuming the simple steady-state theory to apply to an accelerated short-rotor, the copper loss at speed *v* is given by $F(v_s - v)$. If *m* is the mass of the rotor, its acceleration is F/m and the energy dissipated in the rotor in accelerating from rest to synchronous speed is therefore

$$\int F(v_s - v)dt = m\int_0^{v_s}(v_s - v)dv = \tfrac{1}{2}mv_s^2$$

Likewise, the energy put into the rotor copper in decelerating from a speed v_s to rest is

$$m\int_{-v_s}^{0}(v_s - v)dv = \frac{3}{2}mv_s^2$$

The total energy loss for one traverse is therefore four times the kinetic energy at the centre. Rotor dissipation is one of the obvious problems of the unrestrained self-oscillating motor. This loss of energy in the rotor can be considerably reduced by the introduction of a pair of return springs, one fitted at each end of the track. The rotor is designed to have a stable amplitude much greater than the track length, so that it rebounds from each spring in turn. The bulk of the rotor kinetic energy is thus returned to the system instead of being converted entirely into heat as it is in the unrestrained case. The portions of the speed/force characteristic then being utilized are those as for example between the points MM′ and NN′ in Fig. 11(*b*). The overall characteristics of the active part of the motor now resemble very closely those of a single phase induction motor whose rotor may be regarded as being acted on by a positive and a negative torque. The comparison may be taken further

by saying that the same criterion $\tau\omega > 1$ governs whether or not the machine runs at all.

There is no reason why the principle of the end springs should not be applied to a linear single-phase motor which is capable of providing drive force always in the same direction as the velocity. The end springs in this case serve to reverse the velocity. The differences between such a motor and a 3-phase back-to-back system with end springs are negligible so far as efficiency, rotor dissipation and available power output are concerned. The advantage of the 3-phase system, however, is that it is self-starting for all positions of the rotor except the central one.

(3.3) Applications of the Self-Oscillating Motor

Although the self-oscillating motor was developed with a view to shuttle propulsion the problem of adapting the device to existing types of loom is very difficult. The electromagnetic shuttle really demands a complete redesign of the loom, entailing a great deal of development work.[11] Only in the case of the very wide looms is there a possibility of adapting the system to existing machinery. Nevertheless the possibilities of a completely new type of all-electric loom using an electromagnetic shuttle-propelling device are considerable. One revolutionary feature which may be possible is that of a 'floating shuttle' suspended by the alternating field in the gap. A piece of conducting material placed on an open-sided track tends to be lifted off the track as well as propelled along it. The stability of the suspension can be studied by using a single-phase track with the conductor at standstill. Flat sheets cannot be freely supported in this way; the sheets tend to tilt towards the position of minimum energy, and remain supported as shown in Fig. 12(*a*).

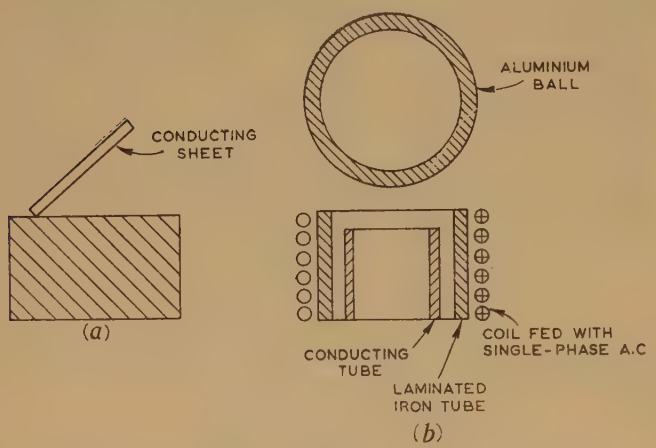


Fig. 12.—Suspension of conductors in alternating fields.

An aluminium ball can be freely supported by an arrangement shown in section in Fig. 12(*b*), where stabilizing is effected by means of a shading ring inside and concentric with the main coil. Double-sided tracks in which the stator coils assist in driving flux across the gap produce the opposite of levitation, a slab of conductor in the gap being attracted strongly by either upper or lower half, whichever is the nearer, just as if the slab were of iron. It is possible that some design of stator and rotor can be found, possibly incorporating shading rings, which will enable a non-ferrous rotor to be suspended on the stator leakage flux. Reference 12 deals more fully with the problem of free suspension.

The rotor can certainly be prevented from leaving the track sideways by means of longitudinal slots in the stators, as shown in Fig. 13. This mechanism has been found satisfactory in practice.

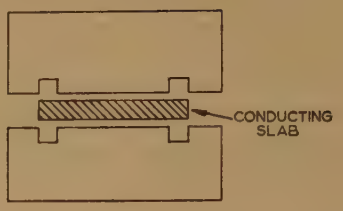


Fig. 13.—End view of double-sided stator with lateral stabilizing slots.

A second application for the self-oscillating motor in the textile industry is under investigation at the present time. This consists of a traverse mechanism used in winding packages of yarn. The demands of the system are a virtually constant-speed traverse with rapid reversal at each end. Amplitudes between 3 and 12 in are normally required. The maximum speeds attainable with mechanical devices are limited to about 750 traverses per minute.

This application calls for the back-to-back system with end stops, and an experimental machine has been constructed which achieves over 900 traverses per minute on a 4½ in traverse. The pole pitch is 1½ in and therefore only 1½ pole pitches are used

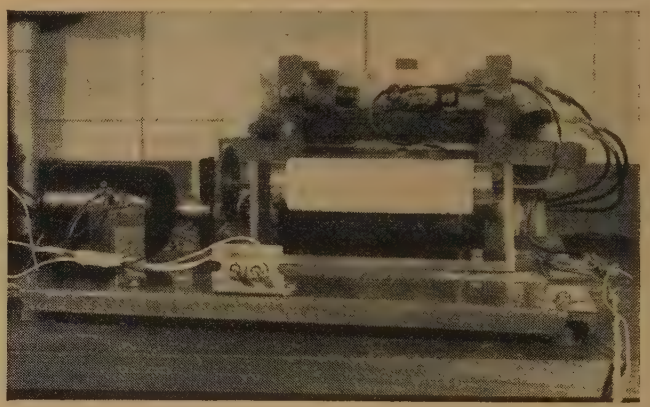


Fig. 14.—Prototype double-sided self-oscillating motor for use as a winding traverse mechanism.

on each side of the centre. The oscillation is stable under these conditions. Fig. 14 shows the traverse motor with a test-recording drum in position.

(3.4) Liquid-Metal Pumps

The high conductivity of liquid metals such as sodium enables them to be pumped by utilizing the pressure set up in the liquid when carrying current in the presence of magnetic field. Liquid-metal pumps provide the rare example of linear motors other than those of the pure induction type, and pumps have been constructed which operate as d.c. machines or as a.c. commutator motors. A survey of the various types of pump has been given by Blake.[13] The induction-type pumps are divided into three classes and given the names Flip, Alip and Sip.

Flip, an abbreviation of 'flat linear induction pump', is a double-sided short-stator machine carrying a rectangular tube in the air-gap down which the liquid metal flows. The side walls of the tube are made of copper to act as high-conductivity end rings.

Alip, standing for 'annular linear induction pump', is a tubular short-stator motor of the type described in Section 2.6 and illustrated in Fig. 9. The liquid flows in the annular space between the stator-tube walls and a cylindrical iron core.

Sip, an abbreviation of 'spiral induction pump', has a stator

constructed in the manner of a conventional rotary machine. This produces a rotating magnetic field. The liquid metal is constrained to move between spiral guide vanes which cause it to pass through the stator as it were along a screw thread. This type of pump cannot therefore be classed as a 'linear' motor, although the net result on the liquid is a linear translation and the motor does, in fact, fall into the short-stator class of machine.

The fact that the requirement for pumping liquid metal arises in connection with nuclear power and radioactive substances places some restriction on the form of construction. It is advantageous to permit the winding to be removed without removing the pipework, and to this extent the flat linear pump seems to be the simplest form of construction. Although all types of pump are essentially short-stator machines, the general practice has been to parallel-connect the windings and employ end grading, rather than to series-connect and allow the flux to be automatically graded. It must be noted that the very nature of the liquid-metal pump and its purpose make it a very different machine from a normal linear induction motor or arch motor. Flow rates of up to 10000 gal per minute, requiring 525 h.p., have to be considered, so that the cross-sectional area of the flow channel required is considerable. Effective air-gaps of the order of 1 in are common in this type of motor, the pole pitch being of the order of 6 in. The resistivity of the liquid being pumped is generally many times that of copper so that the product $\rho_r g$ is large and the rotor time-constant may be 20 or 30 times less than that of a well-designed conventional induction motor. Efficiency and power factor are therefore unavoidably low in liquid-metal pumps. The efficiency is rarely better than 50% overall, and power factors between 40 and 50% are quite common. Stator exit edge loss and excess rotor copper loss, which cannot be controlled because of the small rotor time-constant, together with electrical losses in the tube walls and hydraulic loss, all contribute to the low efficiency. Nevertheless the induction pump is a highly successful device where large powers and low-resistivity fluids are concerned.

(3.5) Further Applications

Table 1 shows the division of linear motors into classified groups. No doubt the Table is incomplete, and the author would be glad to learn of any other applications which have been developed. An interesting possibility which was investigated recently concerned the drawing of thick steel wire. A tubular motor imparting a distributed drive to the wire has certain advantages over pinch rollers at high speed. The fact that the iron-wire rotor would complete the magnetic circuit suggested that a highly successful device might emerge. The drawing process, however, was carried out at a temperature just above the Curie point and the rotor was therefore virtually non-magnetic. The force obtainable per unit length of tubular motor and the efficiency of the device with what was effectively a non-ferrous rotor were both found to be inadequate.

(3.6) Design of Linear Motors

The question of designing short-stator motors in the light of the new theory of transient operation is too complex to be dealt with in a general paper of this kind, and this aspect will, it is hoped, be the subject of a future paper. Short-rotor machines may be designed along more or less conventional lines, it being remembered that the flux distribution near the ends of the stator will require careful consideration to prevent overheating of coils.

An interesting feature of such designs is that the linear motor is in many ways more fundamental than the rotary machine. For example, the number of poles on a rotary machine gives no indication of its order of efficiency. A 20-pole motor, whose rotor diameter is 6 in, is a very poor machine, but a 20-pole motor with a 10 ft-diameter rotor might be expected to have both efficiency and power factor of over 90%. The two factors which effectively fix the shape and size of motors for a given power output and speed are the permeability of iron and its dependence on flux density and the resistivities of copper and aluminium. An economical maximum flux density in the iron having been set, the torque per unit square of surface which can be obtained from a linear motor is dependent only on the current densities in rotor and stator which can be obtained. The limiting factor on current density is the rate at which heat can be taken away so as to limit the temperature rise.

For most conventional machines of reasonable size the working current density is almost the same. The torque per unit square of surface is therefore virtually constant. This is merely stating in terms of a linear machine the well-known formula that the power output is proportional to d^2ln, where d is the diameter and l the length, for the surface velocity, v, of a conventional motor is given by πdn. The power output from a linear machine, per unit square, is proportional only to v, since the torque is constant. Hence the power output for a complete machine is proportional to v times the area of active surface, i.e. to dlv for a rotary machine. Since v is proportional to dn, the output is proportional to d^2ln. It is therefore possible to design rotary machines by making all calculations in terms of a unit square of linear motor. The customer then states the power and speed he would like, and the designer effectively produces one of his standard 'sheets of motor', cuts out a rectangle and rolls it up to meet the requirement. Each different kind of 'sheet' will vary as regards pole pitch, slot depth, and so on, and will be labelled with the leakage and similar quantities per unit square, together with the expected efficiency and power factor, for it is pole pitch primarily which affects the efficiency and the ratio of leakage

Table 1

	Single-sided	Double-sided	Tubular
Short stator		Liquid metal pump Flip Conveyor belt (at present under consideration)	Liquid metal pump Alip Electromagnetic gun
Inverted short stator* ..	Aircraft launching Electropult		
Short rotor	Shuttle propulsion (early schemes)	Shuttle propulsion Traverse mechanism for winding (at present being developed)	
Inverted short rotor ..			

* Machines in which the short-circuited winding is stationary but only partially covered by the polyphase winding.

inductance to coupled inductance. Although such an approach adds nothing new to design theory, it is perhaps a more straightforward technique.

The arch motor is a good example of the application of this principle. If a 200 r.p.m. direct drive of 500 h.p. is required with reasonable efficiency and power factor, the designer looks at his 'sheets of motor' and finds that, for the efficiency and power factor quoted, a pole pitch of at least 4 in is required. The speed demands 30 poles around the periphery. The area of rectangle required for 500 h.p. is found to be 10 ft^2 so that the rectangle is as shown in Fig. 15(*a*) for a conventional machine. The axial

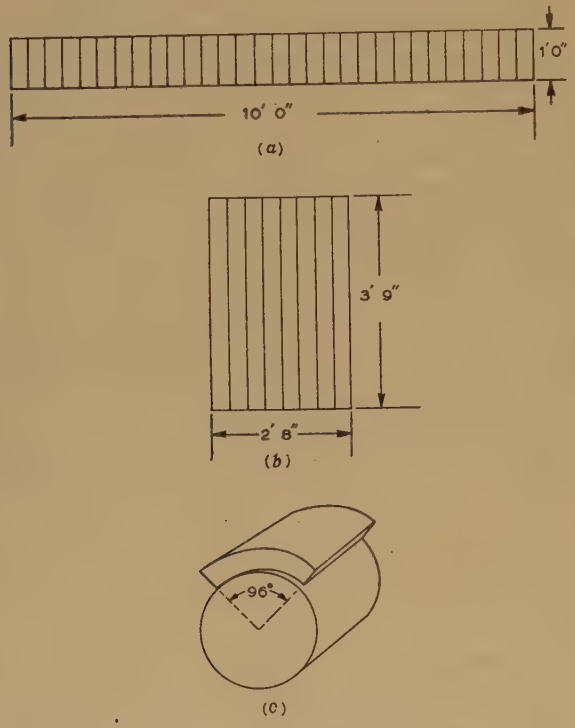


Fig. 15.—Principle of the design of an arch motor.

length of the motor is therefore 1 ft. This is clearly going to entail losses in end windings which are greater than those in the useful part of the winding. The designer therefore cuts his rectangle with 8 poles as shown in Fig. 15(*b*), making it $\frac{30}{8}$ ft wide, and rolls up the piece so that it subtends only $\frac{8}{30} \times 360 = 96°$ at the centre, yielding the result shown in Fig. 15(*c*).

(4) CONCLUSIONS

The fact that a large proportion of motions required in industry by such devices as planing machines, shaping machines, etc., are linear rather than rotational suggests that the linear motor has been hitherto somewhat overlooked. This is not the case, however, simply because the speed of linear motion generally required is too low and the gear is a much more compact device than the lever. The self-oscillating motor certainly offers one or two new possibilities for short-rotor machines, such as driving free-piston compressors, the electrical equivalent of the pneumatic drill, agitators, etc. Conveyor belts may be driven successfully by short-stator machines, provided that the right form of flexible rotor can be found.

Not least of the contributions made by the linear motor has been the opening of a new field of study, that of the transiently operating machine. The full impact of this has yet to be judged.

Table 1 shows most of the applications of linear induction motors so far completed or being developed. Just how many more will be added to such a Table within the next few years is a matter for speculation.

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621.315.051

The Institution of Electrical Engineers
Paper No. 2400 S
Dec. 1957

400 kV TRANSMISSION SYSTEMS IN THE SOVIET UNION

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(*Lecture delivered before the* SUPPLY SECTION, 24*th April*, 1957.)

(1) THE IMPORTANCE OF 400 kV LINES IN THE POWER ECONOMY OF THE U.S.S.R.

The development of the power industry in the Soviet Union, the interconnection of power systems, and the construction of large hydro-electric and thermal stations are all closely associated with the creation of a high-voltage transmission network and inter-system tie lines, and with the problem of increasing the transmitting capacity and the length of transmission lines. The extent and speed with which the Soviet power industry is developing and the enormous area of the country are factors which require the transmission of power over long distances.

After the 220 kV transmission system was put into service in the early 1930's, extensive theoretical, experimental and design work was started to develop the use of a higher voltage of 380–400 kV. This work was interrupted by the war, but was subsequently recommenced with renewed energy. The good results obtained from the work led to the decision in 1949–50 to construct 400 kV long-distance lines to transmit power to Moscow from the large hydro-electric stations being built on the Volga. The construction of the Kuibishev–Moscow transmission line was started in 1952, and the line was put into operation in the spring of 1956. The construction of the Stalingrad–Moscow and Kuibishev–Urals 400 kV transmission lines will be completed in 1958–59.

The development of the power industry (Fig. 1) introduced the problem of creating a consolidated power system in the European part of the U.S.S.R. in the course of the current five-year period. During the first stage of development of this system it will be necessary to transmit energy in one direction from the region of large power resources to load centres with insufficient local power resources. The power, to be transmitted over a distance of 1 000 km, is of the order of $1–1{\cdot}5 \times 10^3$ MW. It is also necessary to build large 400 kV inter-system circuits within the big power systems in the central, southern and Ural regions of the country, and finally to create inter-system tie lines with large transmitting capacities. These problems can be solved by 400 kV transmission lines, each circuit of which should be capable of handling at least 500–750 MW.

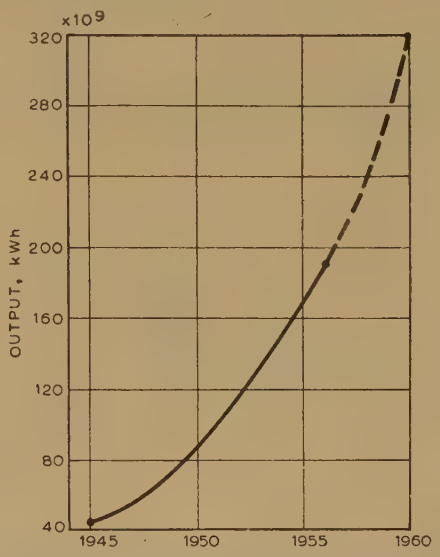


Fig. 1.—Growth in the output of the power stations in the Soviet Union during the post-war years.

In the system of the consolidated power economy the 400 kV lines have several functions, namely

(*a*) To transmit large blocks of power with a capacity factor of 4 500–6 500 hours per year from large hydro-electric and thermal stations to power-deficient areas.
(*b*) To transmit power from one time zone to another.
(*c*) To transmit reserve power in case of equipment repair and fault conditions.

Table 1

DEVELOPMENT OF THE 400 kV TRANSMISSION SYSTEM

Transmission line	Number of circuits	Length	Transmission capacity	Energy transmitted yearly for average hydroelectric conditions	Remarks
		km	MW × 10^3	MWh × 10^6	
Kuibishev–Moscow	2	(*a*) 815 (*b*) 890	1·5	6·0	Put into service in 1956
Stalingrad–Moscow	2	1 000	1·5	8·0	Under construction
Kuibishev–Urals	1	1 050	1·0–0·75	4·0	Under construction
Stalingrad–Donbas	1	500	0·75	4·0	±400 kV d.c. line
Moscow ring	1	215	—	—	Under construction

As a rule, the 400 kV lines carry out all three functions, but for certain lines, such as the Kuibishev–Moscow and Stalingrad–Moscow links, the transmission of large blocks of power in one direction over long distances is characteristic.

The situation as it stood at the beginning of 1957 is shown in Table 1. By 1960 there will be 5 000 km of 3-phase 400 kV line

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constructed in the European part of the U.S.S.R., as well as 500 km of 800 kV d.c. line and 17 substations for the 400 kV lines. The capacity of the consolidated power system will reach 3×10^4 MW in 1960.

The second stage in construction of the 400 kV network in the European part of the Soviet Union will be completed during the seventh 5-year period from 1961 to 1965. A further 6 000–7 000 km of 400 kV line and 25 step-up and step-down 400 kV substations are to be constructed. The capacity of the consolidated power system in the European part of the Soviet Union will be brought up to 5–6 $\times 10^4$ MW by the end of this period (Fig. 2).

Parallel with the development of 400 kV lines in the European part of the U.S.S.R., similar construction is planned for the Asiatic part and during the current 5-year period, the construction of 400 kV lines will be started to connect into one system the power stations in the Irkutsck, Krasnojarsck, Kuznetsck and Novosibirsck districts, which include the hydro-electric stations being built on the Angara (the Irkutsck and Bratsck Stations), on the Enysei (the Krasnojarsck Station) and the big thermal stations in the Kuzbass.

(2) TECHNICAL PROBLEMS OF 400 kV LINES

(2.1) Transmission Scheme

The transmitting capacity of one circuit of a 400 kV line should not be lower than 500–750 MW. Since the economic current density for aluminium–steel conductors is 0·5–0·6 amp/mm^2, the cross-section of the current-carrying aluminium part of the conductor must be at least 1 200–1 600 mm^2 per phase.

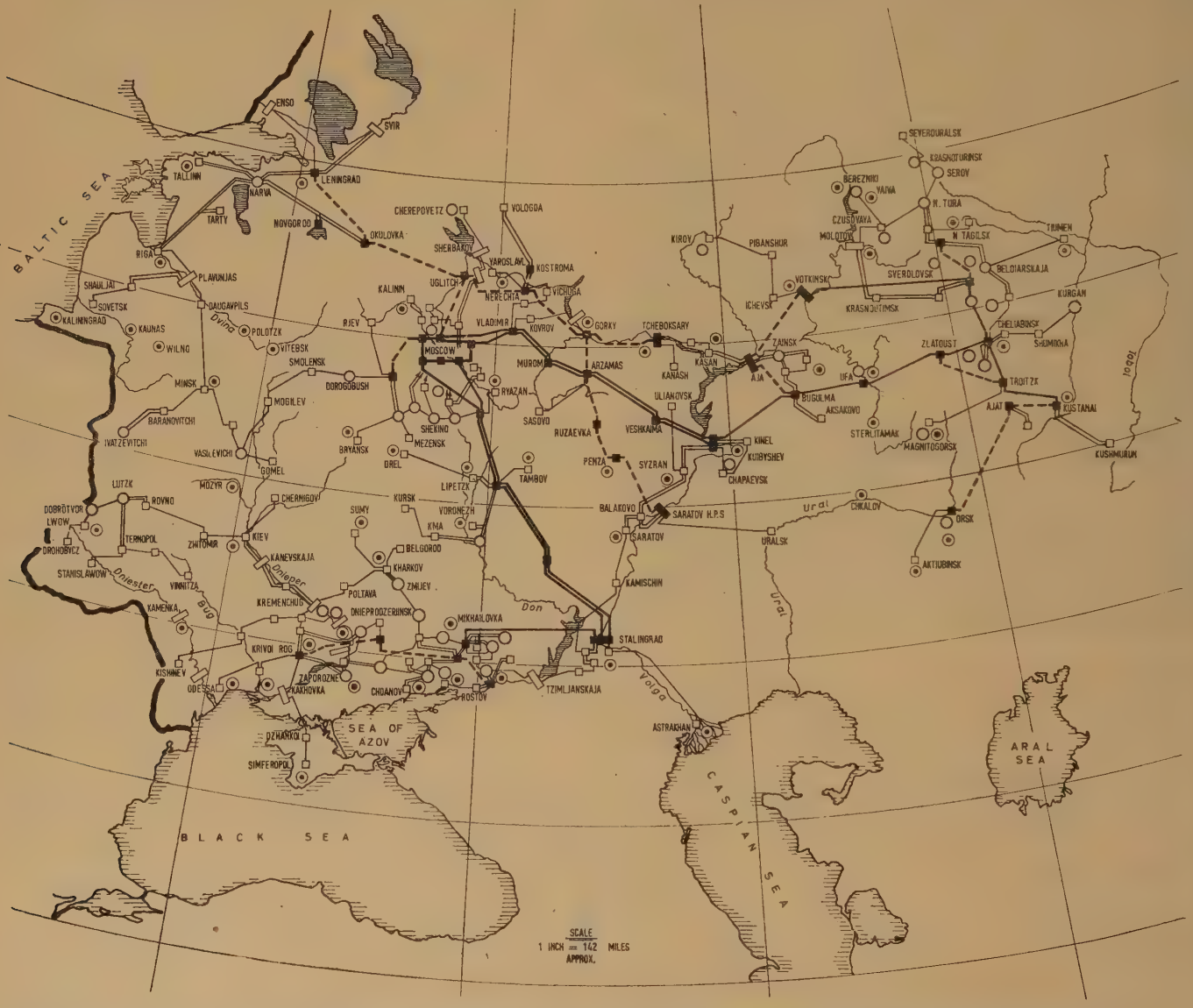


Fig. 2.—The main 400 and 220 kV circuits in the consolidated power system of the European part of the U.S.S.R., as planned for 1965.

- ■ 400 kV substation.
- □ 220 kV substation.
- Hydro-electric station exceeding 150 MW.
- ⊙○ Thermal stations.
- ▬ 400 kV lines to be completed in 1960.
- - - 400 kV lines to be completed in 1965.
- —— 220 kV lines.

NOTE.—Stalingrad–Mikhailovka line is ± 400 kV d.c.

The necessity for keeping the diameter and cross-section of the conductors within practical limits, and of reducing the line reactance, the corona losses and the radio interference, determined the use of bundle conductors. In Soviet practice, the phase bundle is composed of three parallel conductors spaced at the apexes of an equilateral triangle with 400–600 mm sides. For these conditions the cross-section of the aluminium part of a single conductor varies from 400 to 480 mm^2 for the 400 kV lines in service, in construction, and in design.

The most difficult problem encountered in long-distance a.c. transmission is the stability of parallel operation of power stations connected through the long transmission line. Calculations and studies carried out on electro-dynamic system models, and particularly the experience gathered from the first few months of operating the Kuibishev–Moscow line, confirm the possibility of stable operation for a 400 kV line 800–1 000 km in length while transmitting 500–750 MW per circuit if several measures are taken to increase the stability. These are as follow:

(*a*) The use for the synchronous generators of high-speed electronic excitation regulators which react, not only upon a change in the current and voltage, but also on their derivatives (the so-called 'strong action' regulators).

Much operating experience has been gathered from the operation of fast electronic excitation regulators for synchronous machines which were developed in the Soviet Union in the 1930's. If the first types could maintain the transient e.m.f. of the generators constant during transients, the 'strong action' regulators designed for the Kuibishev–Moscow transmission showed during the tests that, as a minimum, they could maintain constant the terminal voltage of the synchronous machine. This circumstance made it possible to raise the initial transmission capacity of the double-circuit Kuibishev–Moscow and Stalingrad–Moscow lines from 1·15 to 1·5 × 10^3 MW each.

(*b*) The design of water-wheel generators for the large hydro-electric stations in accordance with the special technical requirements of reduced transient reactance, increased ceiling value and rate of rise of the excitation, increased inertia constant (this measure is used only for hydro-electric stations connected to the most heavily loaded transmission lines, namely the Stalingrad and Kuibishev stations); the reduction of the reactance of 400 kV power transformers and the use of auto-transformers instead of step-down transformers.

(*c*) The use of bundle conductors for 400 kV lines in order to reduce the line reactance (Figs. 3 and 4).

(*d*) The installation of intermediate switching stations every 200–250 km for sectionalizing double-circuit lines.

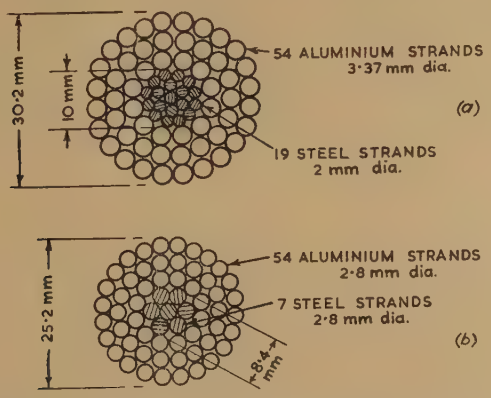


Fig. 3.—Design of conductors used for 400 kV lines.
(*a*) ACSR–ACO480.
(*b*) ACSR–ACO332.

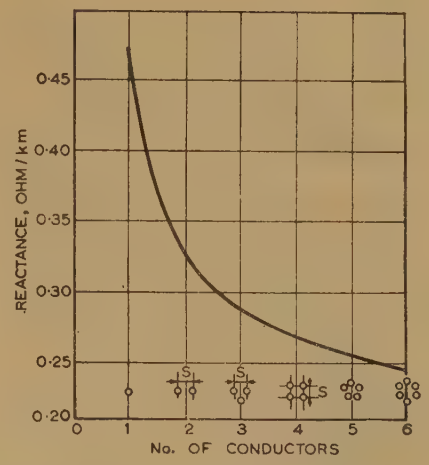


Fig. 4.—The effect of bundle conductors on the reactance of 400 kV lines.
S = 40 cm.

(*e*) The use of series-capacitor compensation. The degree of the compensation must not exceed 40%, because of limitations imposed by the requirements of reducing internal over-voltages (Fig. 5) and increasing the reliability of relay protection. The

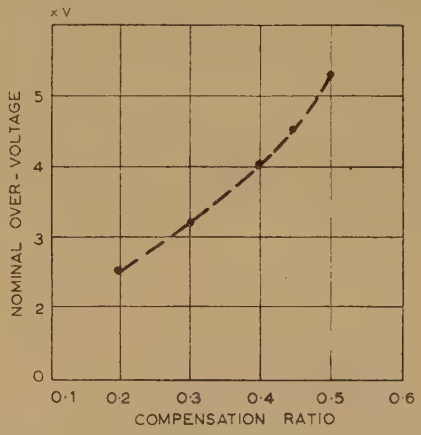


Fig. 5.—The effect of series-capacitor compensation on the level of internal over-voltages in a 400 kV system.

The reactances of the sending and receiving systems are taken into account, as is the increase in impedance of the series-capacitor installation after clearing the faulted section.

The voltage on the capacitors at the moment the fault is cleared is taken as equal to the breakdown voltage of the protective gap across the series-capacitor installation.

good results achieved in testing the 'strong action' electronic excitation regulators, not only permitted the increase in the transmission capacity of the 400 kV Kuibishev–Moscow line, but also allowed the series-capacitor compensation to be reduced from 40 to 33%. The Kuibishev–Urals line is to have series-capacitor compensation of 30%, and the Stalingrad–Moscow line will have 25%. Thus, for the 400 kV lines designed in the Soviet Union there is a tendency to employ capacitor compensation moderately (Fig. 6).

(*f*) The use of synchronous capacitors at the intermediate switching stations along the line route. These condensers are equipped with 'strong action' excitation regulators whose function is to maintain the voltage on the 400 kV busbars of the stations during normal operation of the transmission line, and especially during faults. The question of using synchronous capacitors at the intermediate substations as a means of increasing the stability of long-distance transmission is given special

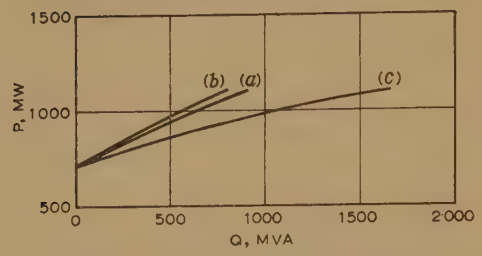


Fig. 6.—The capacity of a 400 kV transmission system 1 000 km long as a function of the compensation rating.

(*a*) Series capacitor.
(*b*) Compensated synchronous capacitor.
(*c*) Uncompensated synchronous capacitor.

attention in the Soviet Union. The installation of two 75 MVAr synchronous capacitors at each switching station is provided for the Stalingrad–Moscow transmission line.

(*g*) The use of high-speed circuit-breakers and relays taking a total of 0·10–0·12 sec to clear faults on the 400 kV line.

(*h*) The installation of load resistors on the 400 kV busbars of large hydro-electric stations, which are automatically switched on to brake the generators in case of sudden load shedding.

Shunt reactors (Fig. 7) are provided to ensure normal operating conditions for the line, and to reduce the losses and the internal over-voltage level. It is expedient under certain load conditions to vary the number of shunt reactors connected so as to regulate the transmission voltage and the flow of reactive power. During the first period of its operation the Kuibishev–Moscow line (Fig. 8) will not have step-down transformers at the intermediate switching stations. An analysis of the problem of where to install the shunt reactors at the step-up substation indicated that it is more advantageous to connect them to the 400 kV side: therefore this transmission line, as well as the Kuibishev–Urals line, has all its shunt reactors designed for

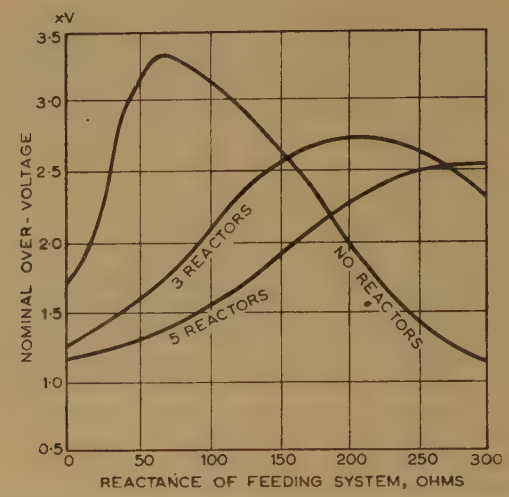


Fig. 7.—The effect of shunt reactors on the internal over-voltage level of a 400 kV system.

installation on the 400 kV side. The conditions for the Stalingrad–Moscow line (Fig. 9) are somewhat different, for three intermediate transformers substations of large capacity will be put into service at the same time as the line itself, and the shunt reactors will be installed on the 110 or 220 kV sides.

The connection diagram, the general layout and the switchgear design for the 400 kV switching stations allow for the possibility of their being expanded into 400/110 or 400/220 kV receiving substations. The time required for converting a switching substation into a step-down substation is determined by the power demand of the adjacent area. In certain cases it is already necessary at the time the line is being constructed to build

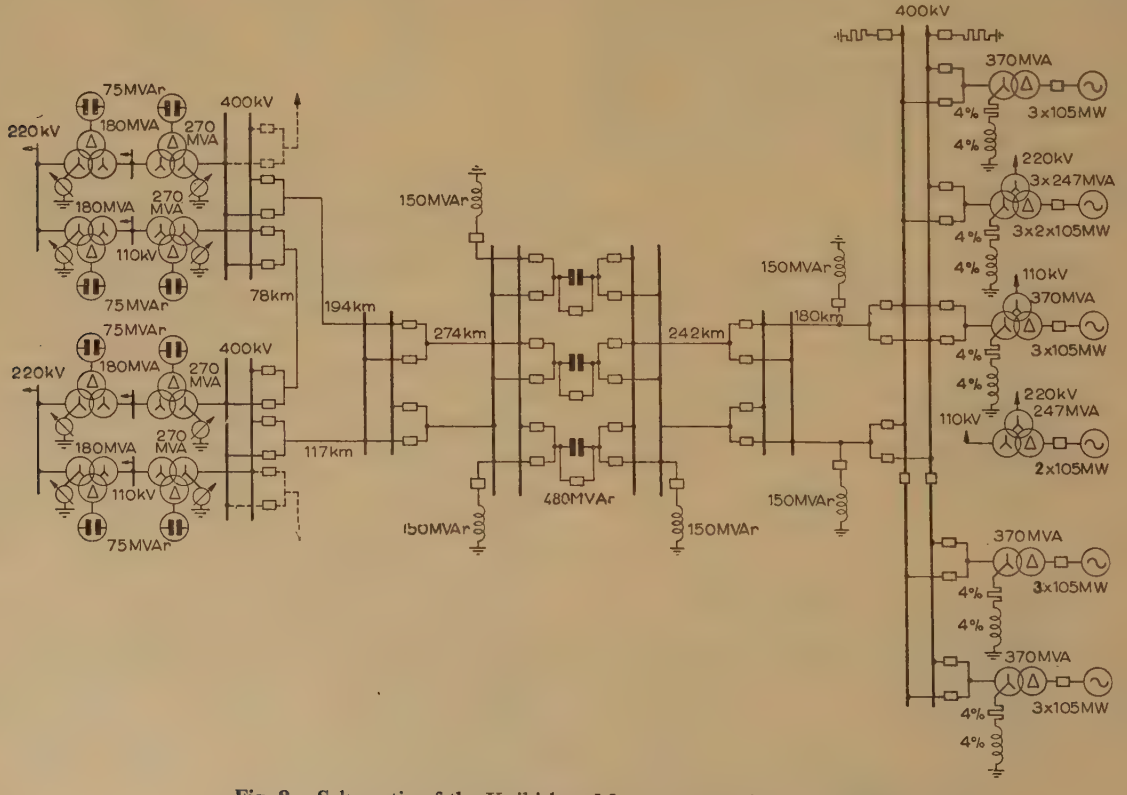


Fig. 8.—Schematic of the Kuibishev–Moscow transmission system.

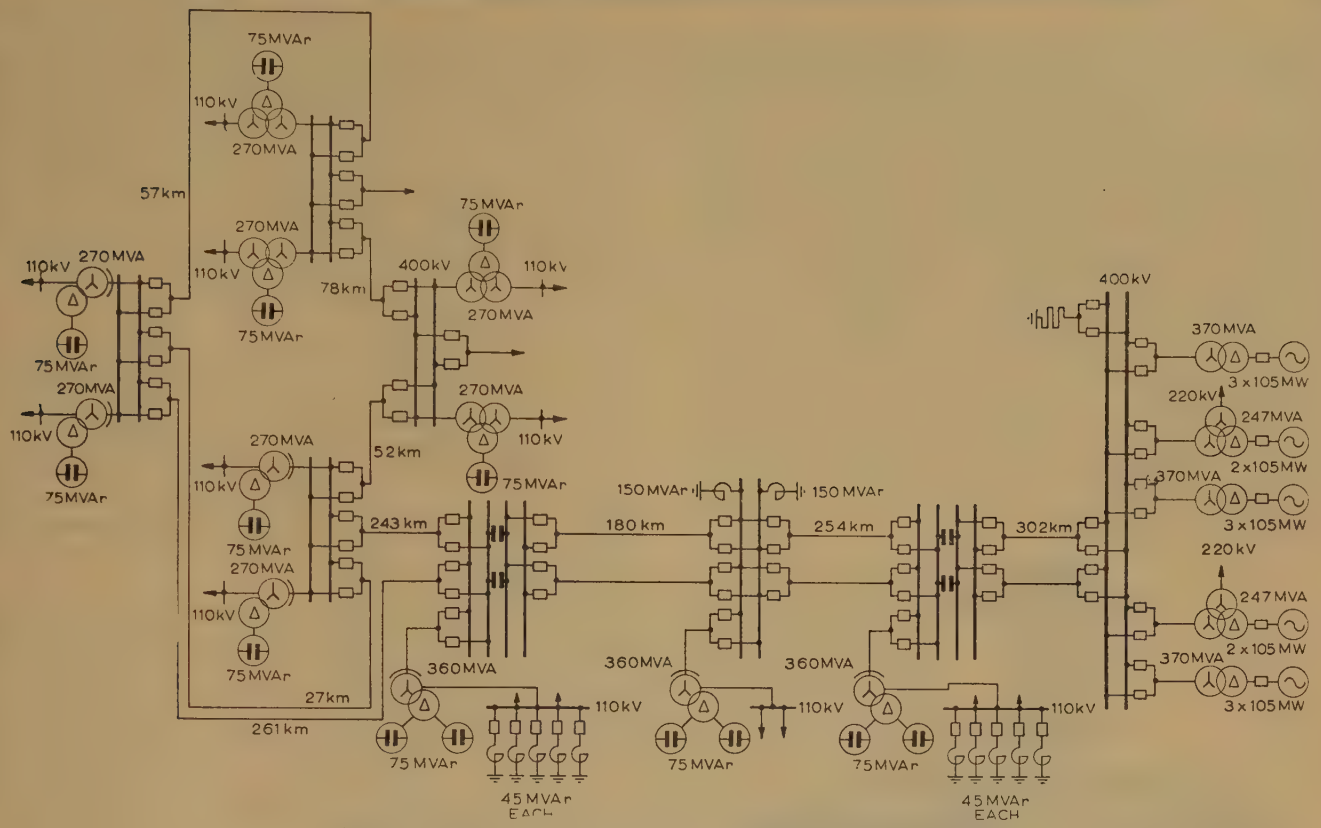


Fig. 9.—Schematic of the 400 kV Stalingrad–Moscow transmission line.

intermediate receiving substations, as on the Stalingrad–Moscow line.

The insulation levels of the 400 kV network are determined by the following conditions:

The neutrals of 400 kV power transformers are solidly earthed.

The 400 kV line is protected along its entire length from direct lightning strokes by two earth wires giving a protection angle of 15–20°.

The footing resistance of the tower is usually up to 10 ohms for normal soil conditions.

The substations are protected from direct lightning strokes by surge-diverters.

The internal over-voltage level does not exceed three times the peak phase voltage.

In accordance with these conditions the following test voltages were adopted:

The 3-shot 1·5/40 microsec full-wave impulse voltage for the line insulation has a peak value of 1·8–2·0 MV, and that for the substation insulation is 1·5 MV.

The 50 c/s wet flashover voltage for the line insulation is 775 kV r.m.s., and that for the external substation insulation is 700 kV r.m.s.

The 1-min 50 c/s withstand voltage for the internal insulation is 700–750 kV r.m.s., and that for the external insulation is 850 kV r.m.s.

(2.2) Transmission Line

One of the most important characteristics of 400 kV lines is that each line must be highly reliable, since its increased capacity and higher cost limit the possibility of constructing double circuits in one direction. It is significant that, in the European part of the Soviet Union, only the Kuibishev–Moscow and Stalingrad–Moscow lines transmitting the largest loads are double-circuit lines; all the other lines being designed and constructed at present are single-circuit lines. The design of support structures is more difficult for lines of this voltage, because of the increased load on the tower arising from the use of bundle conductors, greater electrical clearances, considerable dynamic forces on the tower in case of conductor rupture, etc.

Special technical requirements are imposed on 400 kV line designs which differ from the standards for 110 and 220 kV lines in that certain specifications are made more severe. For example, heavy weather conditions occurring once in 15 years are assumed for the design, instead of those occurring once in five years which are taken for lower voltage lines. The present regulations for 400 kV lines take into account dynamic forces acting on the tower in case of conductor rupture. Requirements for the calculations of metal structures are made more severe; for certain conditions, the allowable tension in the metal of the towers is reduced, etc.

Increased loads on the towers and foundations of 400 kV lines already in the first stage of the design work necessitated seeking ways to reduce these loads. The first step taken in this direction was the use of an aluminium-steel conductor with reduced steel content: the ratio of the steel cross-section to the aluminium cross-section for the 400 kV conductor is 1 : 8, instead of the usual ratio, which lies between 1 : 4·4 and 1 : 5·1 for 110 and 220 kV conductors. A new series of reduced-weight conductors was developed having an aluminium cross-section from 272 to 712 mm^2. This reduced the quantity of steel for the towers and of concrete for the foundations and also reduced the cost of the line. The aluminium-steel conductors have a maximum design stress of 8 kg/mm^2 for the aluminium, the actual safety factor for the reduced-weight conductors being not lower than 2·75.

Reduction of the design loads for the suspension towers was an exceptionally important measure in obtaining an economical design for the 400 kV line, since these towers comprise 90% of all towers used. Suspension towers for all 400 kV lines in the Soviet Union are calculated only on the basis of normal operating conditions when accounting for forces that are present with all the conductors intact. The weight of the wires, insulators and

sleet and wind pressure determine the weight of the suspension tower, which should be equally stable in the longitudinal and transverse directions. There are several ways of eliminating the forces in the suspension tower that arise when conductors rupture, such as the use of releasing clamps or clamps with limited holding strength. In all cases the force on the suspension tower for breaks in all three conductors of the phase bundle should not exceed 1·5–2·0 tons. This is the force at which the releasing mechanism operates; it is also equal to the withstand strength of the element holding the three conductors of the bundle in their clamps. The first 400 kV lines in the U.S.S.R. were equipped with special releasing clamps designed for a phase bundle of three parallel conductors. The releasing mechanism operates selectively: it does not free the wires for breaks in one or two conductors of the phase bundle, since the forces that arise in this case are not dangerous to the tower. However, it definitely frees the wires when all three conductors of one phase bundle break, regardless of the order. The releasing mechanism (Fig. 10) successfully passed tests on the 400 kV test stand and



Fig. 10.—Releasing mechanism for three conductors of a 400 kV line.

functions well on the Kuibishev–Moscow line, never having operated incorrectly.

The disadvantage of releasing clamps is the necessity of having to use strain towers (or more usually angle-strain towers) every 7–10 km to limit the line section in which the wires may fall to the ground when all three conductors of a phase bundle break. In spite of the fact that the probability of all three conductors breaking is quite small, and that such a break may come about only from external causes, the use of releasing clamps requires the installation of strain towers.

A 400 kV line has been designed which allows strain towers to be eliminated. In this, releasing clamps are replaced by clamps having a limited holding strength, so that conductors which break do not fall to the ground but slip in their clamps. This also limits the force applied to the suspension tower by broken conductors. At present, these new clamps are being tested on experimental line sections. The elimination of strain towers reduces the expenditure of metal for towers by 3–4%, and of insulators by 20%. It also considerably reduces the cost of concrete for foundations.

Determining the most advantageous span between towers and height of the towers is very important in designing the line structures. It is also important from the standpoint of obtaining the best possible technical and economic characteristics for the line. Detailed investigations of this question showed that the economical span is 400 m, and the height of the suspension tower to the point where the insulators are hung is 27 m. These figures are pertinent to the European part of the Soviet Union, and also when the common type of ACO 480/60 aluminium-steel conductors are used.

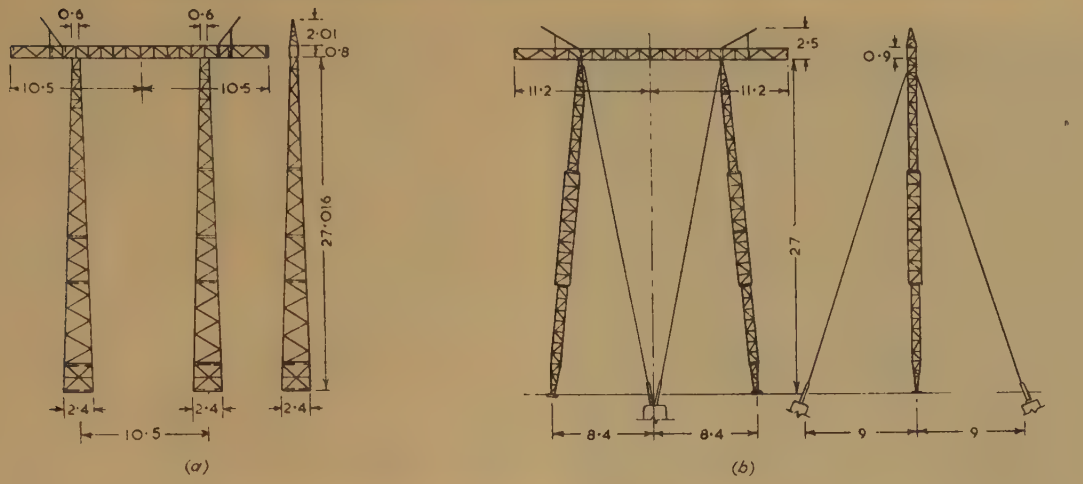
After comparing the design of many types of suspension towers, an H-frame suspension design (Fig. 11) was adopted for the first 400 kV line, Kuibishev–Moscow. This tower has pyramidal posts solidly anchored to their foundations and hinged to the cross-arm. The distance between adjacent phases is 10·5 m. A suspension tower for the Kuibishev–Moscow line, which uses ACO 480/60 conductors, weighs 7·3 tons. Type CT-3 steel is used, having a yield point of 2400 kg/cm^2 and a rupturing strength of 3900 kg/cm^2. The wind velocity adopted in designing the line is 25 m/sec, which is equivalent to a pressure of 40 kg/m^2 along the projection of the conductor and of 55 kg/m^2 on the tower surface. The following foundations are used for the towers:

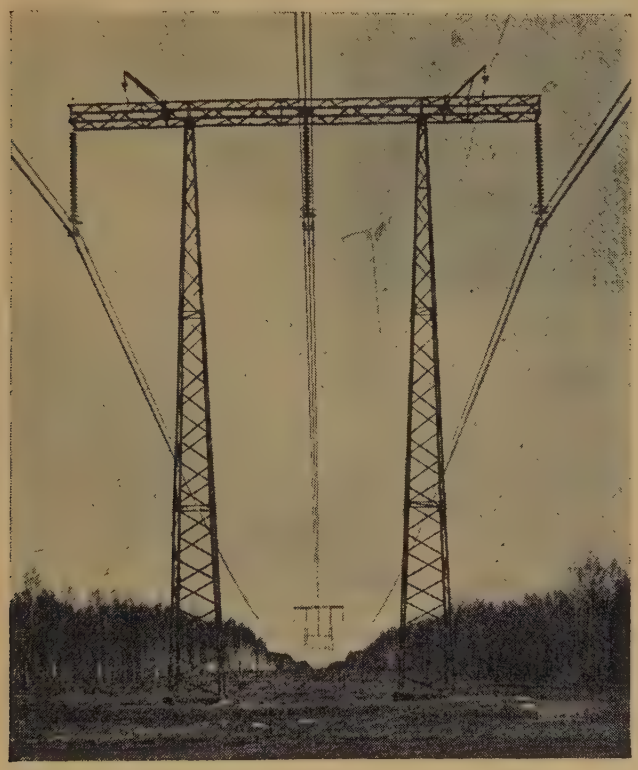
For firm dry soil.—12 m^3 packed foundations (52% of the towers).
For loose soil.—Metal footings with reinforced-concrete slabs (13% of the towers).
For poor soil containing water.—All-concrete massive foundations (35% of the towers).

The same type of suspension tower will be used for the Stalingrad–Moscow line. However, in view of the more severe weather conditions along the route of this line, 90% of which passes through the open steppes, the weight of the tower was increased to 8·6 tons. The design of foundations for the suspension towers in the new lines under construction allows work to be carried out throughout the year. These foundations come either as prefabricated reinforced-concrete mushroom shape-footings (eight footings, each 1·16 m^3) or as reinforced-concrete piles (eight piles, 0·4 m × 0·4 m × 7·0 m, per tower).

For the lines Kuibishev–Urals, Votkinsk Hydro-electric Station–Urals and for the 400 kV lines in Siberia passing through mountainous and forest areas a new type of H-frame suspension tower has been developed; it is supported by steel-wire guys and its frames are hinged to their foundations. These towers save about 16% of the metal, simplify the construction of foundations and reduce their volume from 8–9 m^3 to 3 m^3 per tower. This is an important factor for these regions, where transportation facilities are poor. The use of towers with guys is considered permissible along forest and hilly routes, where there is no danger of the guy wires being damaged by tractors and other agricultural machinery. Twin-circuit 400 kV suspension towers are used only when the route is crowded and each circuit cannot be suspended on a separate tower, since it involves extra metal expenditure for the towers.

The insulators of supporting strings are of the porcelain





(c)

Fig. 11.—Suspension towers for 400 kV lines.
Dimensions in metres.
(*a*) Self-supporting tower.
(*b*) Stayed tower.
(*c*) Self-supporting tower with 400 kV line.

suspension type having an electro-mechanical test loading of 7 or 8·5 tons; there are 20–22 insulators in a string.

The angle-strain towers (Fig. 12) with strain insulator strings are of the bar type for all the lines under construction, since this is the cheapest 400 kV type. Angle-strain towers are installed on solid all-concrete foundations, and where there is good soil, on packed foundations and prefabricated reinforced-concrete footings. Very severe requirements are imposed on the design of these towers, for they must be capable of operating as dead-end towers, and also of taking on the load when two phases (6 conductors) covered with sleet are broken; their weight varies from 23 to 27·5 tons, depending on the angle.

The insulators for the strain strings are made of porcelain, are 320 mm by 200 mm and are tested with an electro-mechanical load of 11·0 tons. The three conductors of the phase bundle are attached to the angle-strain tower through a group of three parallel strain strings each containing 22 insulators.

In the 400 kV lines without strain towers and strings, special

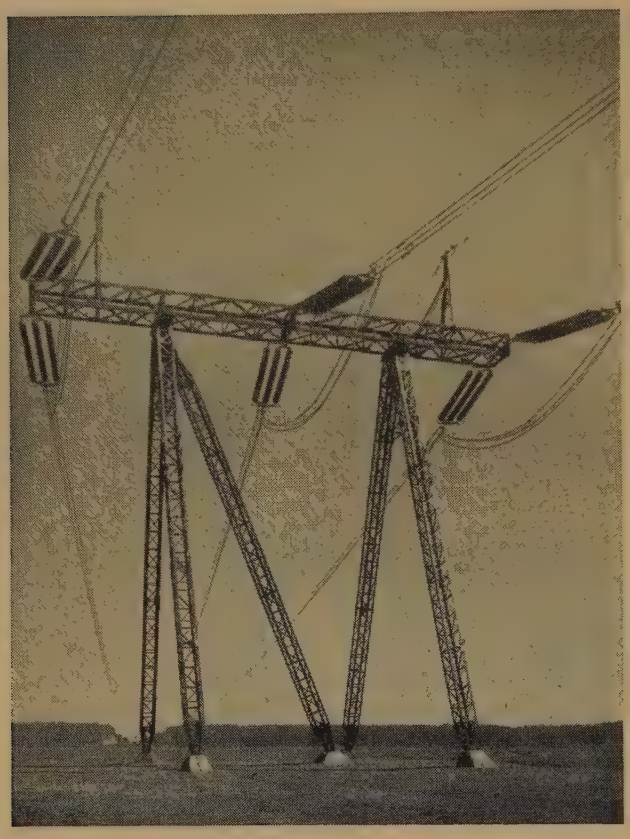


Fig. 12.—Angle-strain tower for 400 kV lines.

H-frame towers have been developed to provide an angle of up to 20°. The conductors are attached to these towers through a suspension string.

The technical and economic features of the 400 kV lines being designed and constructed in the Soviet Union are given in Table 2.

(2.3) 400 kV Substations

The busbar arrangements for the 400 kV step-up, step-down and intermediate sectionalizing substations are primarily determined to ensure reliable operation of the transmission system as a whole and also to reduce the cost of construction. A comparison of several possible schemes from this aspect led to the use of economical but reliable triangular and square schemes as well as the 'transformer busbar' arrangement for the majority of 400 kV installations (with up to 5–6 elements connected). The 400 kV receiving substations (Fig. 13) located near large load centres transform the voltage down to 110 kV and distribute it at this level within a radius of 30–50 km. These substations usually have two transformer or auto-transformer groups rated at 270 MVA each. For a less concentrated load and also for reverse operation of the substation, a transformation ratio of 400/220 kV is used, and one or two auto-transformer groups rated up to 500 MVA are installed at the station. The 400 kV transformers and auto-transformers are always equipped with booster transformers with on-load tap-changing.

In designing the 400 kV outdoor switchgear for the substations of the Kuibishev–Moscow line, special attention was paid to ensure reliable operation of the equipment and to provide for its convenient installation and repair. The switchgear is delivered with metal portal supporting structures and external main busbars. A bay is 28 m wide and a line-bay is 161 m long, the

Table 2.—Technical and Economic Characteristics of Soviet 400 kV Lines in Design and Construction

Line	Number of circuits	Phase cross-section of aluminium-steel conductors	Span	Design conditions		Type of suspension tower	Weight of suspension tower	Type of suspension clamp	Type of foundation for suspension tower	Angle-strain towers		Steel expenditure, including foundations	Characteristic of route	Remarks
				Wind pressure on conductors	Weight of sleet load					Type	Weight			
		mm²	m	kg/m²	kg/m		tons				tons	tons/km		
Kuibishev–Moscow	2	3×480/60	460	40	0·5	H-frame	7·3	Releasing	All concrete	Bar	23·0–27·5	29·5	Level	In operation
			425		1·15									
Kuibishev–Tatnjeft	1	3×480/60	400	46	1·15	H-frame	8·6			Bar	23·0–27·5	28·7	Level	Under construction
Tatnjeft–Zlatowust	1	3×480/60	400	46	1·15	H-frame supported with guys	7·2			Bar	23·0–27·5	27·2	Mountainous	
Stalingrad–Moscow	2	3×480/60	400	46	1·15	H-frame	8·6		Reinforced concrete piles and footings	Bar	23·0–27·5	29·4	Level	±400 kV d.c. line
Stalingrad–Donbas	1	2×712/93	370	57	1·15	Single-post T-frame	4·3			H-frame	13·0	18·3	Level	
Troitsk–Cheljabinsk	1	3×480/60	400	46	1·15	H-frame	8·6			Bar	23·0–27·5	31·0	Level	Design data
Votkinsk–Sverdlovsk	1	3×400/50	390	46	1·15	H-frame supported with guys	7·2			None	—	23·4	Mountainous	
Bratsk–Irkutsck ..	2	3×480/60	415	57	1·15	H-frame supported with guys 50%	7·2	Limited holding strength		None	—	28·0	Mountainous	
						H-frame 50%	8·6							

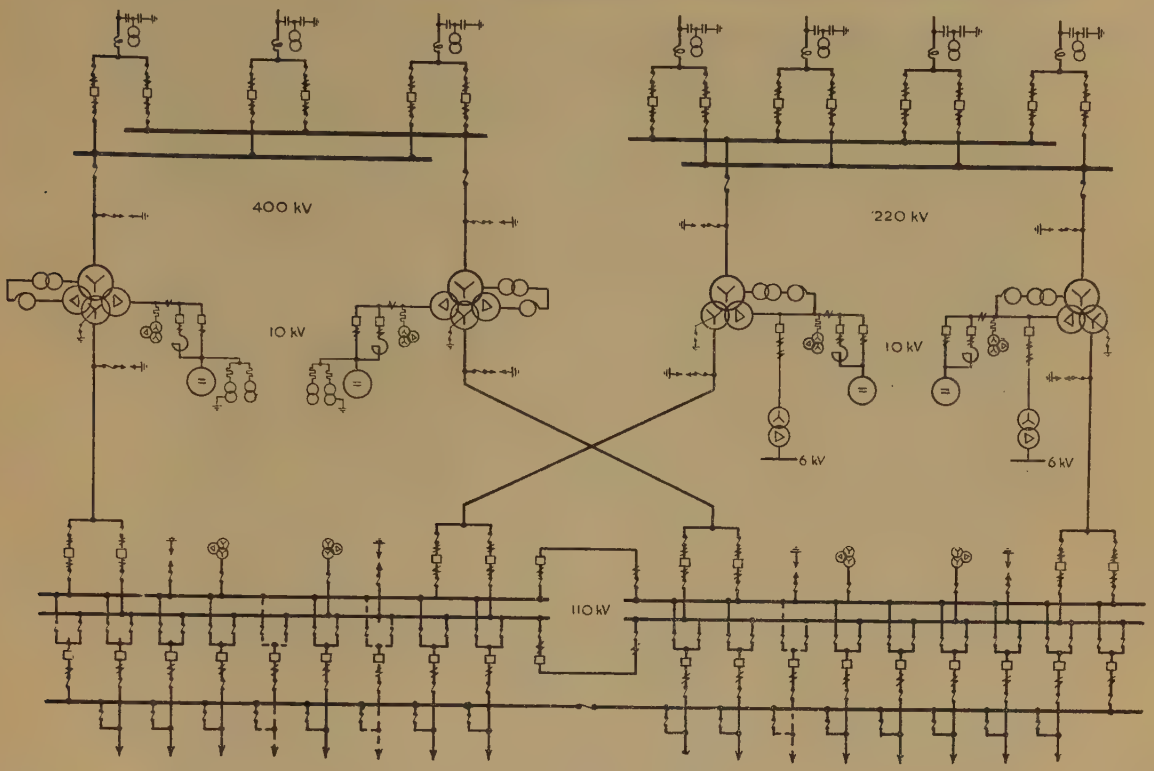


Fig. 13.—Schematic of the 400 kV Northern substation.

distance between phases being 6 m. The busbars for a phase consist of two bare copper conductors, each 300 mm^2 in cross-section, arranged horizontally 400 m from each other.

The design of 400 kV outdoor switchgear (Fig. 14) for the subsequent transmission systems was reviewed and simplified in the light of experience gained from the construction, installation and operation of the first 400 kV substations, and will be used for the 400 kV Stalingrad–Moscow and Kuibishev–Urals transmission systems. It gives a 20% saving in the material for the structures and in the cost of constructing and installing the switchgear.

The arrangement of the 400 kV substation depends on the number of outdoor switchgear units for various voltages, the way the transmission line and the railway approach the station, and also on the terrain in the substation area.

For large substations with three voltages of 400, 220 and 110 kV a frontal arrangement is used (Fig. 15). For substations with only two voltages, 400 and 110 kV or 400 and 220 kV, a closed arrangement is frequently used where one set of switchgear is placed behind the other and the power transformers and synchronous capacitors are installed in the middle along the railway tracks. All the equipment for the 400 kV substations is supplied by the electrical industry of the Soviet Union, and brief characteristics of the main equipment are given below.

Step-up Transformers.—Rating 370 MVA in 3-phase group (3 × 123·5 MVA); voltage 13·8/420 or 13·8/121/420 kV delta-star-star connection; neutral on 400 kV side earthed either solidly or through a low resistance; short-circuit reactances: h.v.–m.v. = 19·5%; h.v.–l.v. = 14·5%, m.v.–l.v. = 5·5%; water cooled with forced oil circulation.

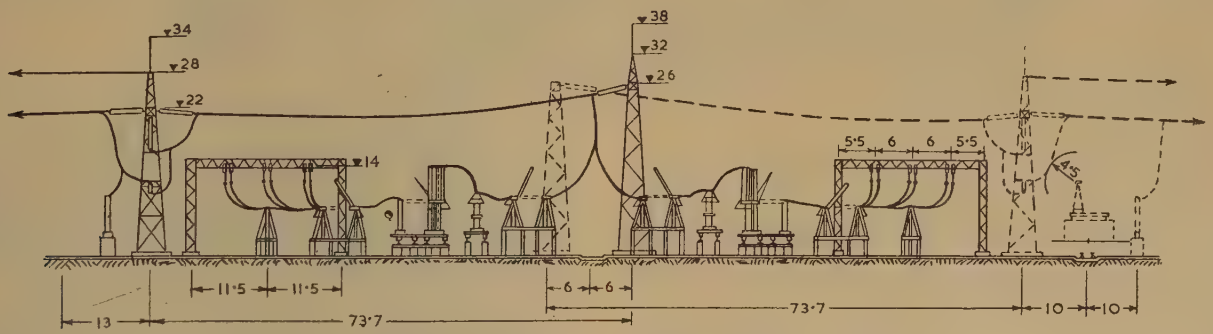


Fig. 14.—Section of the new design of 400 kV outdoor switchgear.
Dimensions in metres.

Step-down Transformers (Fig. 16).—Rating 270 MVA in 3-phase group (3 × 90 MVA); voltage 410/115/11 kV; neutral on 410 kV side solidly grounded; stat-star-delta connection; short-circuit reactances: h.v.–m.v. = 13%; h.v.–l.v. = 19%, m.v.–l.v. = 5%; air cooled.

Step-down Auto-transformers.—(*a*) Rating 270 MVA in 3-phase

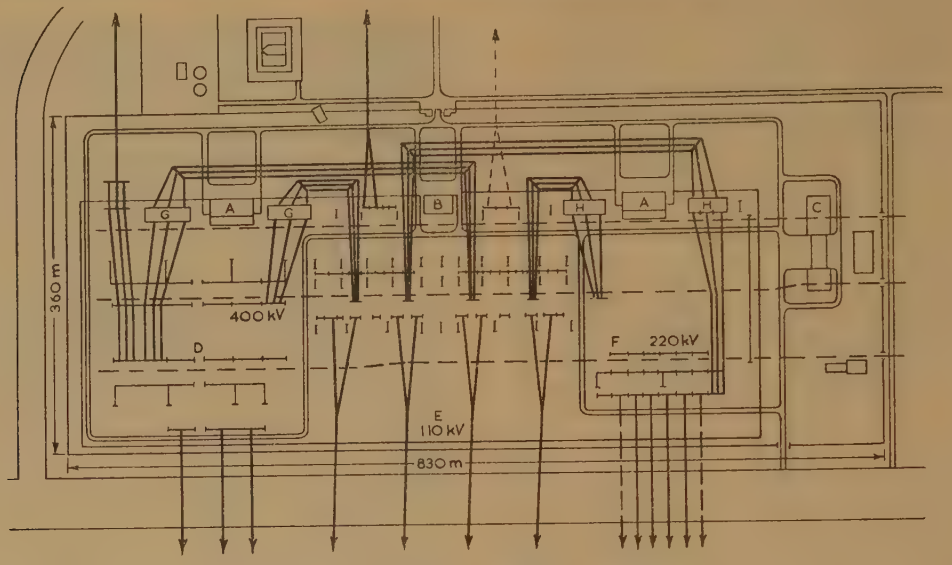


Fig. 15.—Plan for a 400/110/220 kV substation.

A. Building for synchronous condensers, auxiliaries, etc.
B. Main control room.
C. Transformer workshop.
D. 400 kV outdoor switchgear.
E. 110 kV outdoor switchgear.
F. 220 kV outdoor switchgear.
G. 440 kV transformer bank.
H. 220 kV transformer bank.

group (3 × 90 MVA); voltage 410/115/11 kV; neutral on 410 and 115 kV sides solidly earthed; 410/115 kV windings star connected, 11 kV winding delta connected; booster transformer for on-load tap-changing connected in neutral of 410/115 kV windings; short-circuit reactances: h.v.–m.v. = 10·5%, h.v.–l.v. = 19·5%; m.v.–l.v. = 9·0%; air cooled. (*b*) Rating 500 MVA in 3-phase group (3 × 167 kVA); voltage 420/242/11 kV; neutral of 420/220 kV windings solidly earthed; 11 kV winding delta connected; short-circuit reactances: h.v.–m.v. = 10·5%; h.v.–l.v. = 15·5%; m.v.–l.v. = 12·5%; air cooled.

400 kV Circuit-Breakers (Fig. 17).—(*a*) Air blast with disconnector; rated current 2 kA; interrupting capacity of 10 000 MVA; disconnecting time of 3 cycles. (*b*) Similar to (*a*) but with interrupting capacity of 15 000 MVA.

400 kV Disconnecting Switches (Fig. 18).—Knife blade; rated current 2 and 1·5 kA; a.c. motor drive.

Voltage Transformer.—Cascade type, transformation coefficient $(420\,000/\sqrt{3})/(100/\sqrt{3})/100$ volts; power consumption 300 VA; accuracy class, 0·5.

Current Transformer.—Cascade type, transformation coefficient 2 000–100–500/1 amp.

Shunt Reactor.—Single-phase rating of 50 MVA; voltage, 400 kV; neutral solidly earthed; air cooled.

400 kV Coupling Capacitor.—Stack type with capacitance of 6 250 pF.

Carrier-Frequency Trap.—Suspension type; self-inductance of 2 mH, rated current of 1 850 amp.

(3) FIRST RESULTS OF OPERATING 400 kV KUIBISHEV–MOSCOW LINE

The first circuit of the transmission line was constructed by the autumn of 1955, except for the troublesome section in the Djigoulee district. By that time, construction and installation work at the 400/110/220 kV Eastern substation in Noginsk had also been completed. This allowed research work and tests to be carried out on the line during the winter of 1955–56, with the line energized at 400 kV from the Moscow end.

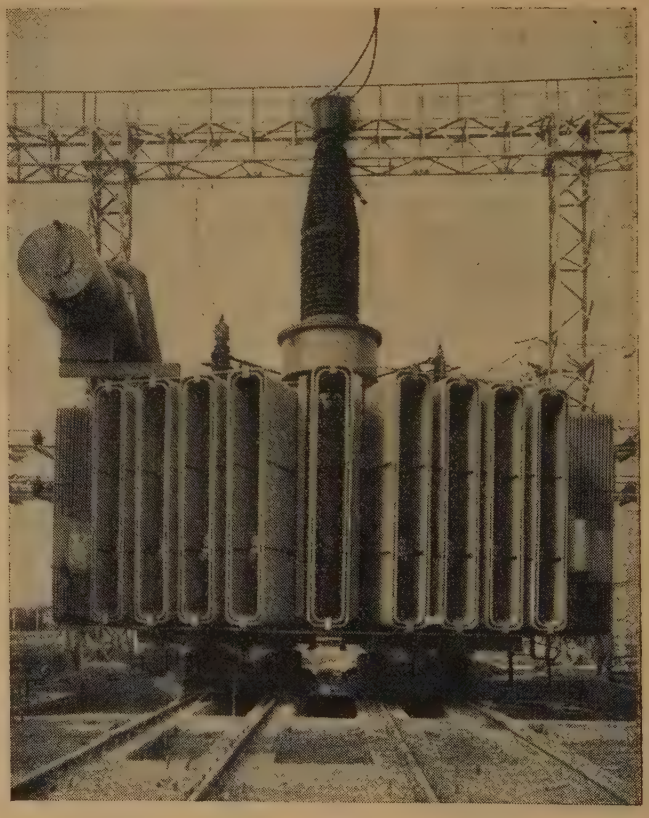


Fig. 16.—400 kV step-down transformer.

The programme of this work included measuring the line parameters, checking the operation of equipment and apparatus, and measuring voltages and currents for different operating conditions of the line. By the spring of 1956 the Djigoulee section

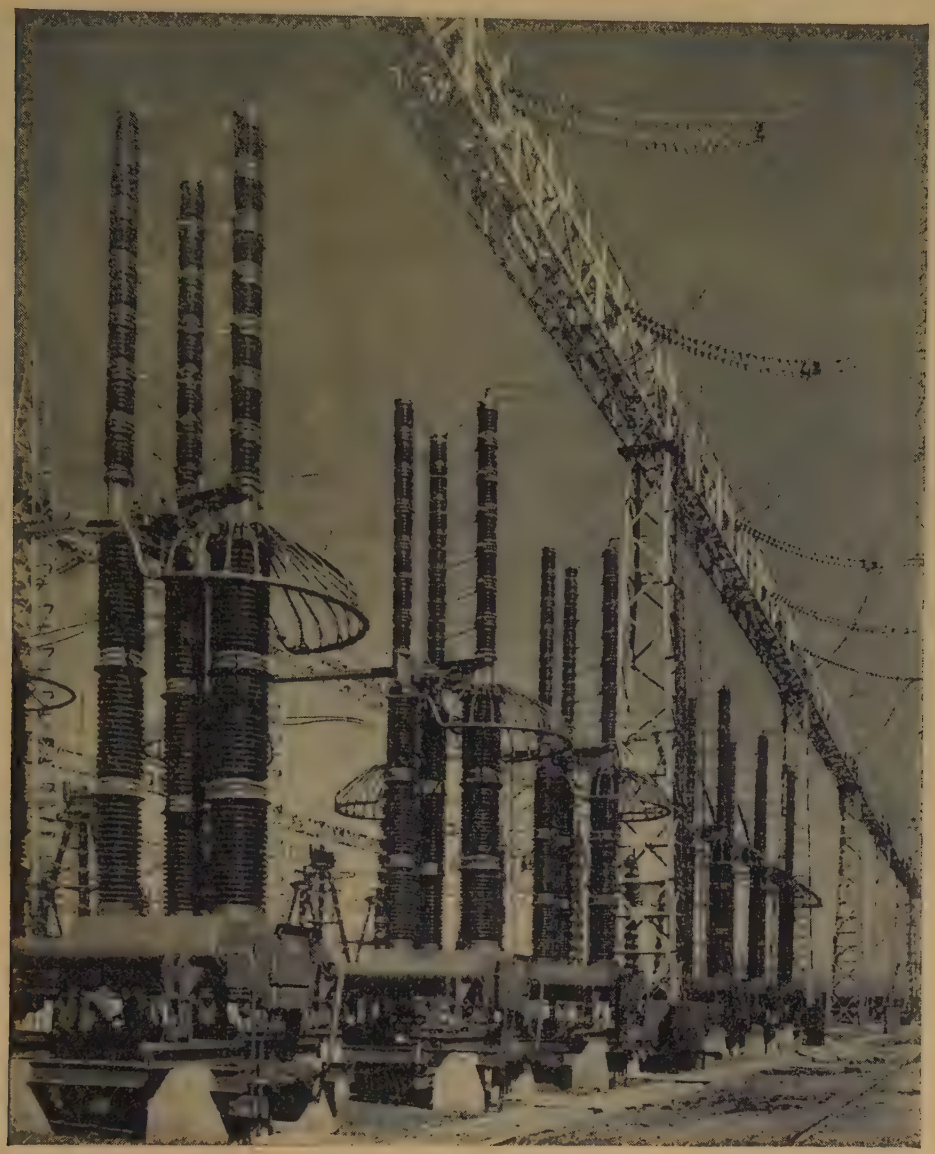


Fig. 17.—400 kV air-blast circuit-breakers.

of the line was constructed and the 400 kV outdoor switchgear at the Kuibishev station was installed. This enabled the test programme to include 2-way feed of the transmission line.

In April, 1956, the necessary tests were made and the first circuit of the 400 kV Kuibishev–Moscow line, 815 km long, was put into service. In October, 1956, the second 400 kV substation at the Moscow end was put into service as well as the 77 km 400 kV line between Noginsk and Northern substation. Early in of December, 1956, the second circuit of the transmission line was put into service.

During eight months in 1956 the 400 kV line transmitted $1\,750 \times 10^3$ MWh from Kuibishev to Moscow, the maximum power transmitted through one circuit being 500–520 MW. After the second circuit was put into service in December, 1956, the transmission capacity increased, and in February, 1957, over 800 MW was transmitted over the two circuits.

Actual system tests of the transfer capacity of one circuit of transmission line from the aspect of steady-state stability showed that, without series-capacitor compensation and with only standard excitation regulators on the generators, the stability limit for a voltage level of 420 kV is 560–580 MW. With a steady-state stability factor of 10–15%, the transfer capacity of one circuit under these conditions is 500–470 MW.

The line parameters were determined experimentally in the course of this test programme. The measured values of the phase-to-earth, C_{11}, and phase-to-phase, C_{12}, capacitances per unit length of line being 0·008 78 and 0·001 03 μF/km, respectively; the working capacitance, $C_w = C_{11} + 3C_{12}$, is 0·011 87 μF/km.

The frequency characteristics of the line resistance and reactance were also determined between 10 and 130 c/s, and the results are given in Fig. 19. In this frequency range the positive-sequence resistance increases by 1·5 times, while the zero-sequence resistance increases by 8–10 times. The measured values of the positive-sequence resistance and inductance, as well as the line capacitance at 50 c/s, are sufficiently close to those assumed in the design, but the measured values of the zero-sequence parameters, especially the resistance, differ considerably from the design values which were calculated by Carson's formula (see Table 3).

While the line was being tested, corona losses were measured on the line sections from Eastern substation to switching stations

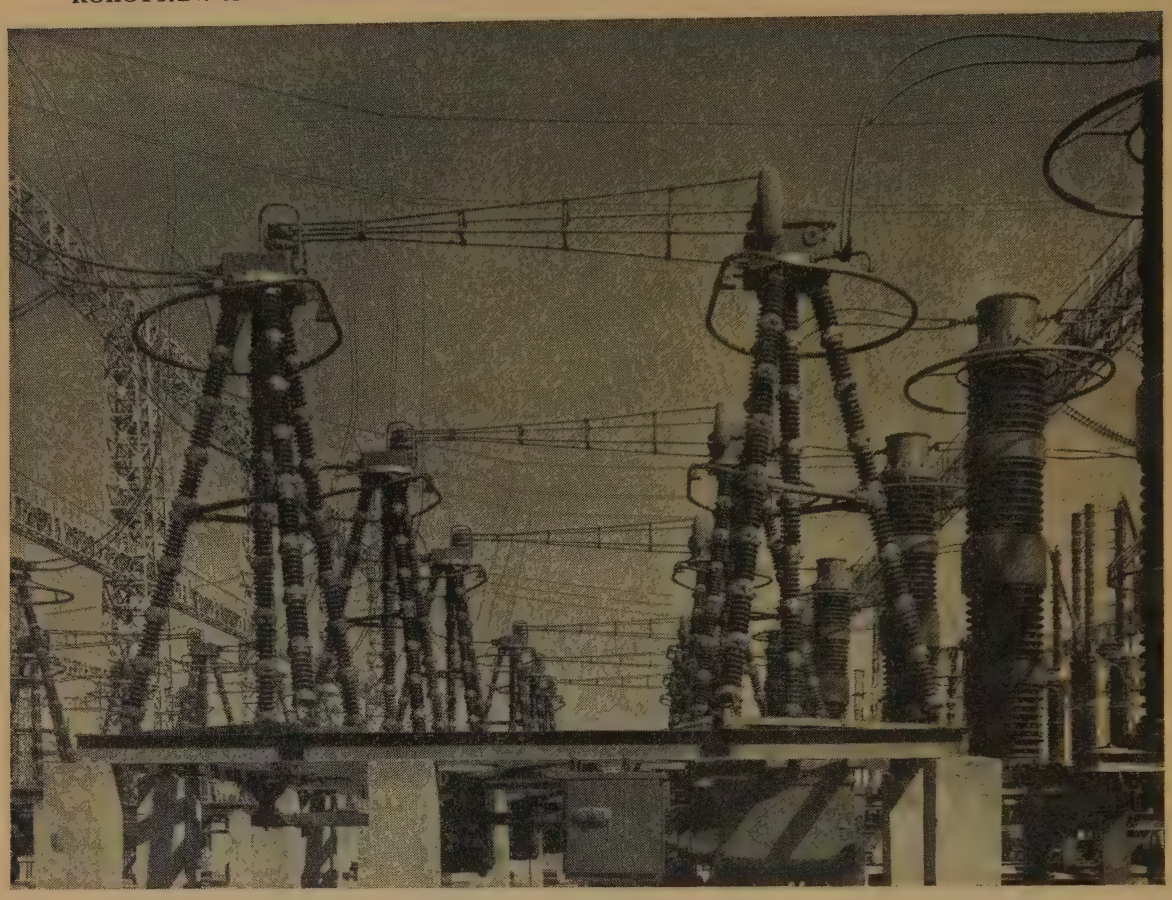


Fig. 18.—400 kV disconnecting switches.

Table 3

CHARACTERISTICS OF 400 kV LINE

Parameter	Test value	Design value
C_{11}, μF/km	0·008 78	0·008 67
C_w, μF/km	0·011 87	0·012 50
R_1, ohm/km	0·023 6	0·022 0
L_1, mH/km	0·946	0·948
R_0 for 117 km, ohm/km	0·365	0·171
R_0 for 391 km, ohm/km	0·257	
L_0 for 117 km, mH/km	3·48	2·97
L_0 for 391 km, mH/km	3·46	

Table 4

BUSBAR VOLTAGES ON 400 kV LINE

Operating conditions	Voltage on 400 kV busbars	
	Switching station No. 2	Kuibishev hydro-electric station
391 km section connected at Moscow end, without reactors	480–510	—
The same, with 150 MVAr of reactors connected to switching station No. 2	400	—
815 km section connected at Moscow end, without reactors		800
The same, with 150 MVAr of reactors connected to switching station No. 2 at Kuibishev hydro-electric station	420	415

Nos. 2 (391 km) and 3 (117 km). The maximum losses during hoar frosts measured on these sections were 15–16 kW/km for the three phases, while on the test line in Leningrad this figure amounted to 18 kW/km for similar weather conditions. The losses on the line in clear weather were 3 kW/km for the three phases, while on the test line they were 0·6 kW/km. The discrepancy is explained by the fact that during measurement the conductors on the Kuibishev–Moscow line were in the process of ageing—a process which takes 6–12 months.

The voltage was measured at various points along the unloaded transmission line (Table 4), the supply being connected to various lengths of open line from the Moscow end with different numbers of shunt reactors connected.

The line is usually connected for operation at the Moscow end with two reactors connected to the line; then the Kuibishev hydro-electric station is synchronized.

The measurement of internal over-voltages for the block transmission scheme of one circuit without series compensation is of interest. Results of the most severe tests are as follows: with 635 km of open line, conditions were created whereby the voltage at the beginning of the line before the circuit-breaker cleared was 1·5 times the phase voltage, V_p. The line with a reactor at the middle switching station was connected to the Moscow end, and the reactor was then disconnected. This increased the voltage throughout the line. After a short time, when the voltage at the beginning of the line reached $1{\cdot}5V_p$ the open line was cleared by an air-blast circuit-breaker at the

Moscow end. The voltage across the circuit-breaker contacts in disconnecting the open line for the above-mentioned conditions reached $2 \cdot 6 V_p$. The circuit-breaker did not arc back.

The largest over-voltages were measured when disconnecting the transmission line with a minimum number of generators connected for a single line-to-earth fault on the Eastern sub-

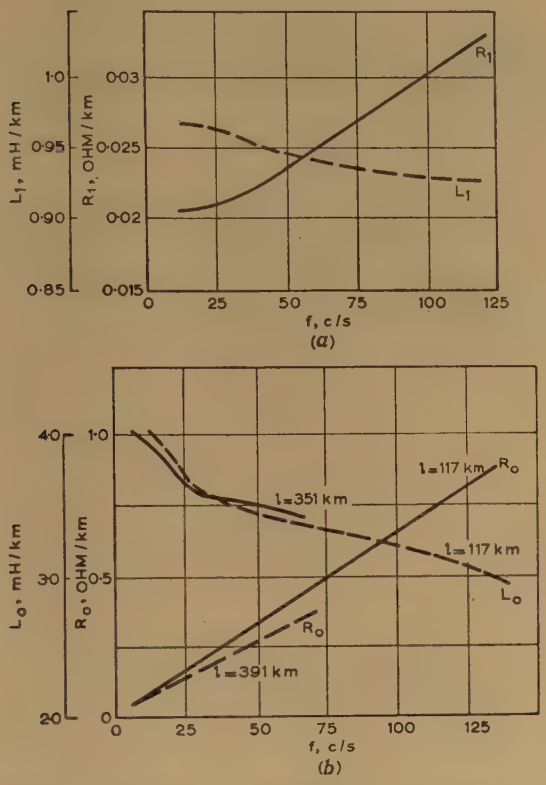


Fig. 19.—Frequency characteristics of a 400 kV line.

——— Resistance.
— — — Inductance.
(*a*) Positive sequence.
(*b*) Zero sequence.

station 400 kV busbars. This fault was cleared by open-circuiting 391 km of line with the air-blast circuit-breakers at the middle switching station (see Fig. 20).

Five such tests were conducted, the results of two, in which the largest over-voltages were measured, being given in Table 5: per-unit voltages are given, the base being 420 kV. The first circuit-breaker to clear was B_3, at the Eastern substation, and then B_5, at the middle switching station. For the above conditions the air-blast circuit-breaker at the middle switching station successfully cleared the line without arcing back.

In the course of the tests, 14 faults were imposed. Three were solid faults and five were arcing faults to check the action of the relay protection. The carrier-current relays and single-phase automatic reclosing functioned properly. Self-synchronizing tests for the generators at the Kuibishev hydro-electric station were also conducted. When the generators had less than 50% of their rated load they fell into synchronism instantaneously. For loads exceeding 50% rated, they pulled into synchronism after slipping for about 16–17 sec, and current surges during the self-synchronizing process did not exceed 300% of the rated current. Tests were conducted to study the clearing of an arcing earth fault when disconnecting only the faulted phase. Many such faults were imposed on line sections from 117 to 815 km long. The results were quite positive, confirming the possibility of using single-phase automatically reclosing circuit-breakers for 400 kV line sections up to 500 km in length.

Studies of the carrier-current channels along the 400 kV conductors indicated that the anticipated interference levels were exaggerated, the actual levels being 2·5 nepers below their design values.

During the summer of 1956 there were very many thunderstorms along the line route. Nevertheless, the line was never disconnected on this account.

Since 1953 galloping conductors were noticed eight times. In one case, all three phases of a span galloped with an amplitude of 7–8 m, and in another case, one phase of a span galloped with an amplitude of 2–4 m. One phase of the transmission across the Ustinski reservoir, 1 080 m long, galloped with an amplitude of 4–5 m.

After changing over in 1955 from the earlier designed chequer-

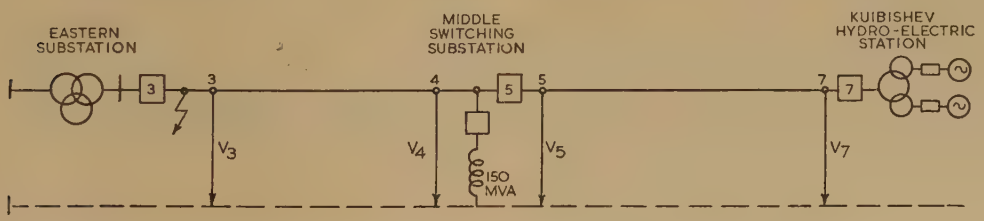


Fig. 20.—Schematic for testing an air-blast circuit-breaker on interruption of transmission.

Table 5

Condition	Measured per-unit voltage when disconnecting circuit-breaker B_3 at the Eastern station						Measured per-unit voltage when disconnecting breaker B_5 afterwards at the middle switching station			
	On contacts of B_3		At point U_3		At point U_4		On contacts of B_5		At point U_5	
	Peak	Steady-state	Peak	Steady-state	Peak	Steady-state	Peak	Steady-state	Peak	Steady-state
Line cleared without single line-to-earth fault	1·55	1·22	2·20	1·82	2·10	1·78	2·55	1·28	1·95	1·40
Line cleared with single line-to-earth fault at point 3 near Eastern station busbars	1·80	1·30	2·60	2·10	2·50	2·0	3·30	1·40	2·00	1·40

board arrangement for the spacers to the group arrangement, no defects in operation were noticed.

Studies show that the dampers, which are to prevent the wires from vibrating, entirely serve their function. The vibration protection scheme for long spans also justified itself.

The insulation and line accessories did not show any signs of failures or defects during tests on the line.

On the whole, the 400 kV lines 815 and 890 km long were put into service quite normally. The good performance of the line in 1956 confirms the proposals made by Soviet engineers in working out the design in 1950–52.

(4) CONCLUSION

The problems to be encountered in the future in the field of large high-voltage a.c. transmission systems are as follows:

(*a*) To improve the solutions to the problem already put forward, and to reduce the cost of 400 kV transmission systems.

(*b*) To solve problems connected with converting the 400 kV lines already in service for operation at 500 kV, since it is possible to reduce the internal over-voltage; thus the insulation of the installed equipment will be more fully utilized.

(*c*) To develop the theory of using a still higher a.c. voltage of 600–750 kV; to carry out the necessary research and to design equipment for this voltage level that will provide for a further increase in the capacity and length of the transmission system.

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DISCUSSION ON 'A NEW METER FOR THE KILOVOLT-AMPERE-DEMAND CHARGE'*

WESTERN CENTRE, AT BRISTOL, 11TH FEBRUARY, 1957

Mr. W. H. Campbell: In referring to the use of metal rectifiers in the meter, the author acknowledges that slight variations in forward resistance and reverse leakage with current and temperature are to be expected, but assures us that the rectifiers are so liberally rated as to make such variations negligible. Would he care to comment on the selection of rectifiers and control of the materials during manufacture, and on the possibility of deterioration or change in characteristics with age even when operated well within the design voltage? What length of useful life free from excessive testing and maintenance can be expected from the meter in this country, having regard to the six years' operating experience so far gained under various climatic conditions?

Mr. R. W. Maltby: In view of the large fluorescent-lighting load of some installations resulting in the consumer drawing a non-sinusoidal current from the supply, I am surprised to note that no reference is made, in the paper, to the accuracy of the meter under conditions of waveform distortion. The conventional type of maximum-demand meter operating on the induction-vane principle is responsive to the r.m.s. value of the current, this value being of importance in assessing the load capacity of the capital equipment. The meter described in the paper rectifies the current from the auxiliary current transformers and passes this current through the control windings of the transductor—the output of which will now be a measure of the mean value of the load current, calibrated in terms of r.m.s. value. The author's experiences in this respect will be appreciated.

Mr. P. Baxter (*in reply*): The interest shown by Mr. Campbell in rectifier selection and the control of materials is shared by many who are responsible for measurement and control circuits which include metal-plate rectifiers.

Rectifier makers apply meticulous care in the control and application of materials used in the manufacture of selenium rectifier plates, and although each batch produced may be satisfactory for most applications, only a percentage will be entirely suitable for measurement purposes, and these are selected during final processing and testing.

After completion of each rectifier, acceptance tests are applied to each plate and repeated just prior to assembly into equipment or meters.

The most significant test rejects any rectifier having a reverse or leakage current which is unstable or in excess of a value predetermined during type testing.

Any new type of rectifier must be subject to shelf-life and load tests over periods of several months to ensure stability.

Selected in this manner selenium-plate rectifiers are as reliable and permanent as other electrical components, and normal maintenance routine need not be altered. References 8 and 9, and experiences in this country and the tropics over the last six years, confirm this.

Mr. Maltby has touched upon a subject the complexities of which the present paper could not embrace.

All meters for assessing a kilovolt-ampere maximum-demand charge are affected. Most types are responsive to the fundamental only, but some operate according to the total r.m.s. value and others according to the average value of the current. All are calibrated with current of sinusoidal waveform.

Tests carried out on a 3-phase 4-wire supply in which the load contained 7% third, 7% fifth and 13% seventh harmonics showed that differences between meter types could be of the order of 2%. Whichever type is selected it is difficult to pronounce judgment, and expediency must of necessity replace exactitude if the load current of a particular consumer deviates greatly from the sinusoidal form.

* BAXTER, P.: Paper No. 1927 M, November, 1955 (see 103 A, p. 263).

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The Institution of Electrical Engineers
Paper No. 2354 S.
Apr. 1957
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BRITISH COLUMBIA–VANCOUVER ISLAND 138 kV SUBMARINE POWER CABLE

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(*The paper was first received 20th November, 1956, in revised form 28th January, and in final form 13th February, 1957. It was published in April, 1957, and was read before* THE INSTITUTION *25th April, 1957.*)

SUMMARY

The paper describes the background and salient features of the 138 kV a.c. submarine cable interconnection linking the lower mainland of British Columbia with Vancouver Island.

After a description of the electrical systems of the British Columbia Electric Co., Ltd., serving these areas, the requirements for additional power are formulated and the considerations leading to the adoption of a submarine cable interconnection are discussed. A following Section explains the determination of the cable route by hydrographic surveys, and the influence of physical site conditions on design and installation. The cable interconnection circuits and associated control, communication and protection systems are briefly described.

The considerations resulting in the adoption of the pre-impregnated gas-filled type of cable manufactured in continuous lengths up to 15·6 miles, are then discussed, together with the development, design and performance characteristics of the cable and accessories; and the development, design, and installation of the special plant necessary for the manufacture and handling of the cable in continuous lengths are reviewed.

The transport and installation of the cables and accessories are described in the following Section, and the methods and reason for landing the shore ends by a special scow are discussed. The paper concludes with a brief note on the field testing, commissioning and operating record of the cables.

(1) INTRODUCTION

On the 25th September, 1956, five submarine power cables, linking the lower mainland of British Columbia with the southern portion of Vancouver Island, were put into service. This brought to a close four years of planning, engineering and construction to meet the rising electrical needs of the territory served by the British Columbia Electric Co., Ltd. (B.C.E. Co.) on Vancouver Island. The background and novel features of the project merit description in view of the widespread interest in the transmission of power over long distances by means of submarine cables.

(2) THE BRITISH COLUMBIA ELECTRIC COMPANY SUPPLY SYSTEM

The B.C.E. Co. serves the south-western mainland of British Columbia as well as the southern portion of Vancouver Island, which is separated from the mainland by the Strait of Georgia—an overall distance of about 30 miles. The location and extent of the service areas are shown in Fig. 1, which also indicates the principal generating and transmission facilities. The installed generating capacity of the mainland system amounts to 524 MW at present, with an additional 540 MW under construction. The one-hour peak capability of the Southern Vancouver Island system is 76·9 MW, as detailed in Table 1. This includes a contract for 37 MW of purchased power, which may be terminated in 1959.

Table 1

B.C.E. Co.–VANCOUVER ISLAND SYSTEM
One-Hour Peak Capability

Source	Type	Peak capability
		MW
Jordan River plant	Hydro-electric	26·4
Diversion plant ..	Hydro-electric	1·5
Brentwood plant ..	Steam (oil-fired)	12·0
Purchased power ..		37·0
		76·9

On the mainland of British Columbia, it is estimated that undeveloped hydro-electric resources exceed 15000 MW continuous. By contrast, the area served by the Company on Vancouver Island has meagre water-power potential, and contains no further hydro-electric power sites suitable for development. Moreover, no additional purchased power is available on a long-term basis from another undertaking which serves the northern portion of Vancouver Island. While the latter territory still contains some undeveloped water power, it is becoming apparent that the remaining economic hydro-electric sites will have to be devoted to the rising electrical needs of Northern Vancouver Island.

During the past seven years, the Vancouver Island system of the B.C.E. Co. experienced a persistent load growth in excess of 10% compound per annum, and a continuation of this trend is expected for some years to come. The actual and predicted one-hour peak loads of the system are shown in Fig. 2. During the winter of 1955–56 the one-hour peak load reached 75 MW and practically equalled the system peaking capability. This situation had been anticipated on the basis of load studies made several years ago, when it became evident that an additional source of power would have to be provided during 1956.

There was also a need for additional peaking capacity from the standpoint of system reserve, since all four units at the Jordan River hydro-electric plant are served by a single wood flume, making them vulnerable to simultaneous outages; in addition, the Brentwood steam plant contains some 4 MW of obsolescent and inefficient units. Reference to Fig. 2 indicates that, with the possible termination of the purchased power contracts, about 120 MW of additional power will be required over the next six or seven years to provide the necessary peaking capability and system reserve.

(3) ALTERNATIVE METHODS OF SUPPLY TO VANCOUVER ISLAND

In the absence of additional sources of hydro-electric or purchased power, consideration was given to a conventional thermal plant, an atomic power plant, and a submarine cable interconnec-

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Mr. Gibson is with British Insulated Callender's Construction Co., Ltd.



Fig. 1.—Principal facilities of the British Columbia Electric Co., Ltd., as at 1st January, 1957.

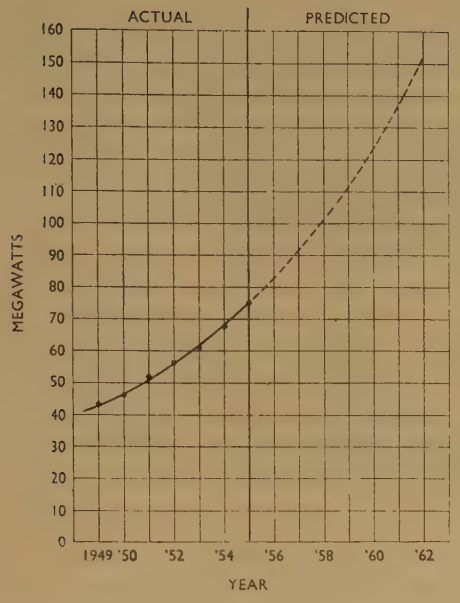


Fig. 2.—British Columbia Electric Company–Vancouver Island system.

Actual and predicted one-hour peak loads.

tion for transmitting hydro-electric power from the B.C.E. Co. mainland system.

It was estimated that the cost of power from a modern 120 MW steam plant with two units, operating at 60% capacity factor and burning Bunker C oil or coal, would be between 0·7 and 0·8 cent per kWh (0·62–0·71d.), including fixed charges, operation, labour, maintenance and delivered fuel cost.

An atomic power plant of 120 MW capacity was not available commercially for service in 1956. Furthermore, estimates indicated that atomic power would be considerably more expensive than power from a conventional steam plant, at least for some time to come.

The unit cost of transmission over a 120 MW submarine cable interconnection was estimated at slightly above 0·1 cent per kWh (0·089d.) when operating at 60% capacity factor. Using hydro-electric power from the mainland system, the total cost of power delivered at the Stratford switching station on Vancouver Island via the submarine cable interconnection would be lower than the cost of power from a local steam plant. The unit cost of transmission over the interconnection would embrace operation, maintenance and fixed charges on five submarine cables (two of which would be spares), chases and landing facilities for seven cables, and provisions for expansion at the substations. Ultimately, there would be seven submarine cables—three for each circuit and one spare—thereby doubling the capacity of the interconnection to 240 MW and further lowering the unit cost of transmission.

Apart from the economic advantages, other benefits would accrue from an interconnection between the mainland and Vancouver Island systems. The hydro-electric plants on the southern portion of Vancouver Island are subject to the run-off characteristics of one watershed; in a dry year there is the danger of an energy shortage, and in a wet year water may have to be spilled. An interconnection with the mainland system would improve the co-ordinated use of several watersheds having diverse run-off characteristics, resulting in more efficient use of the Company's water resources. Also one block of reserve capacity for emergency conditions would suffice for both systems.

It was evident, therefore, that an interconnection by high-voltage submarine power cable would be very attractive. After detailed consideration of the many aspects involved, particularly those which lay outside the field of existing practice and experience, it was concluded that the submarine cable interconnection was feasible, and a decision was taken to proceed with it, based on a 138 kV 60 c/s system, to link up with the existing system.

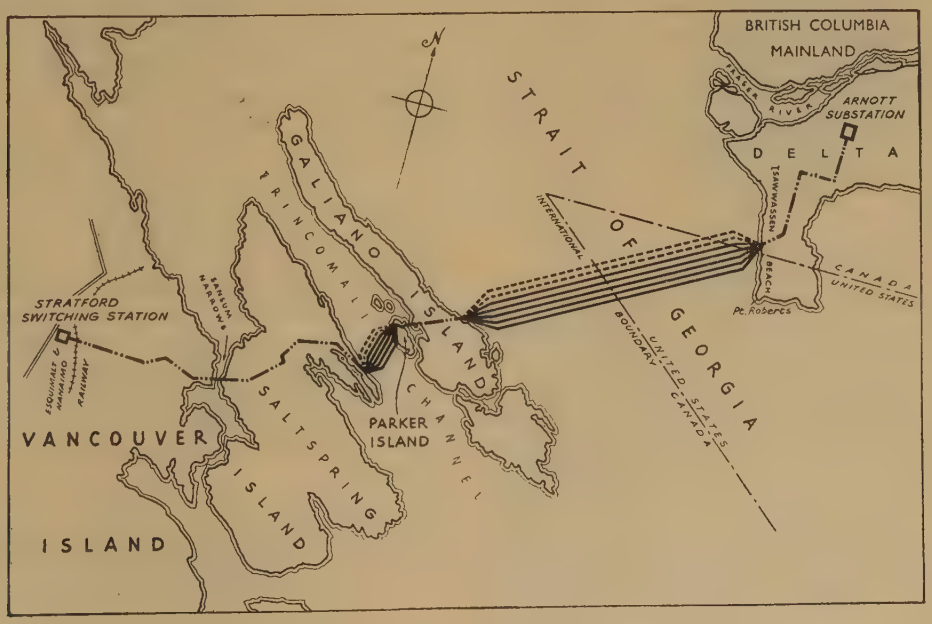


Fig. 3.—Mainland–Vancouver Island interconnection route.

——— Existing cable.
– – – – – – Projected cable.
·—·—· Overhead lines.

(4) HYDROGRAPHIC SURVEY, SITE CONDITIONS AND ROUTE SELECTED

(4.1) Hydrographic Survey

A map of the general area traversed by the cable is shown in Fig. 3. Hydrographic surveys were carried out, soundings and accurate sea-bed contours were taken, bottom currents were measured, and bottom samples were collected. A profile of the route finally adopted is shown in Fig. 4.

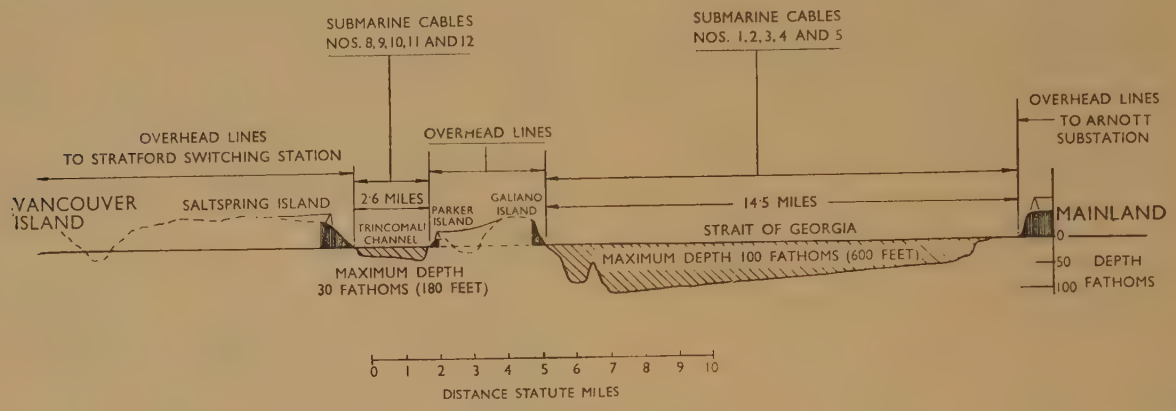


Fig. 4.—Profile of route.

The body of water separating Vancouver Island from the mainland contains a large number of long and narrow islands rising steeply from the sea bed. Strong tidal currents of up to 4 knots at the surface are common. Bottom conditions are partly sand and gravel, and partly rock, pitted with deep holes and channels having abrupt contour changes. Fortunately, a portion of this rugged sea bottom was found to be covered by silt deposited from the mouth of the Fraser River. This part of the Strait of Georgia offered a reasonably smooth and soft bottom as far as Galiano Island, except for a ridge of rock parallel to and about 2 miles from shore. It was therefore decided to land the cables at Galiano Island.

The channels between Galiano, Parker, Saltspring and Vancouver Islands are much less attractive from the cable-laying point of view. However, a detailed survey of Trincomali Channel between Parker and Saltspring Islands revealed a relatively small portion of bottom which was satisfactory, and short 2·9-mile submarine cable sections are therefore employed for this part of the route. For the mile-wide Sansum Narrows between Saltspring and Vancouver Islands, it was decided to utilize an aerial crossing. The general submarine conditions are summarized in Table 2.

Table 2

SUMMARY OF SUBMARINE CONDITIONS

Item	Strait of Georgia	Trincomali Channel
Length of cable section	14·7 miles	2·9 miles
Maximum depth	100 fathoms (600 feet)	30 fathoms
Bottom conditions	Generally sediment, but with a ridge of rock parallel to and at a distance of 2 miles from shore	Generally sediment
Peak of ridge	Ridge rises from 100 fathoms to 40 fathoms at the peak, and falls to 80 fathoms on the Galiano Island side	
Currents ..	Surface: 4 knots average Bottom: 1 knot maximum	
Corrosion ..	No corrosion conditions due to organic or chemical sources	
Marine life ..	No teredo present	

(4.2) Hazards from Shipping

The route of the cables and the landing sites were selected so as not to provide such good anchorages that the cables would be in constant danger of damage from ships' anchors. The Strait of Georgia is a main shipping lane for deep-sea, local and ferry traffic to coastal points of British Columbia and Alaska. In 1955 over 32000 vessels called at the Port of Vancouver alone. In addition there is a great deal of tug, log-boom and barge traffic close inshore, as well as heavy seasonal salmon fishing by fleets of up to 5000 Canadian and American vessels. With the co-operation of the Department of Fisheries, beam trawling was prohibited in Trincomali Channel to prevent damage to the cables; but no trawling restrictions were imposed in the Strait of Georgia.

(4.3) Route Selected

The total length of the interconnection is 43 miles, and the route finally adopted is shown in Fig. 3. Starting at Arnott substation of the B.C.E. Co. mainland system, it runs by way of an overhead transmission line to a point near the International Boundary at Tsawwassen Beach. Here a 14·7-mile section of submarine cable crosses the Strait of Georgia to Galiano Island, running down to a depth of 600ft. It should be noted that, with the permission of the American authorities, the cables take the shortest route across the Strait through United States territorial waters for a distance of about seven miles.

From Galiano Island a connecting overhead transmission line crosses over to Parker Island, where a further 2·9-mile section of submarine cable runs under Trincomali Channel to Saltspring Island. Another section of overhead transmission line then traverses Saltspring Island and spans Sansum Narrows, which is over a mile wide; and the line proceeds to Stratford terminal station, where it links up with the B.C.E. Co.–Vancouver Island system.

(5) CIRCUITS, PROTECTION, CONTROL AND COMMUNICATION SYSTEMS

(5.1) Cable Circuits

The 138kV overhead-line sections have four conductors and connect to four submarine cables. Three cables are used at any one time for the 3 phases of the interconnection, while the fourth circuit can be switched at the terminal stations as a spare for any

phase. A fifth cable serves as standby at present, and provision has been made for an additional two cables at a future date. Ultimately there will thus be seven cables, providing two 120 MW 3-phase interconnections with one common spare circuit.

(5.2) Protection and Control

Both terminal stations are unattended and controlled by supervisory carrier-current equipment from the load-dispatch centres in Vancouver and Victoria, respectively. Two-zone impedance relays are used for phase protection, and high-speed directional earth-impedance relays for earth protection. A single-zone impedance back-up relaying scheme is provided to guard against failure of the primary relays and circuit-breakers. In the event of a phase-to-earth fault, the circuit-breakers at the terminal stations are tripped, and the spare circuit is automatically substituted for the faulted phase, using motor-operated switches. The mainland circuit-breaker is then reclosed, and the Vancouver Island system is resynchronized, either manually or automatically, from the load-dispatch centre in Victoria.

(5.3) Communication and Alarms

Communication and gas-pressure alarm facilities are provided through the co-operation of the British Columbia Telephone Company. Low-gas-pressure alarms are signalled at the telephone exchanges closest to the cable terminals. An audible alarm is given, together with two signal lights for gas failure and one light for circuit failure. Alarm conditions are reported by the exchanges to the B.C.E. Co. load-dispatch centres.

(6) DEVELOPMENT AND DESIGN OF CABLE

(6.1) General

The basic problems associated with the cable design were mechanical rather than electrical. It was considered that the cable must be manufactured in continuous lengths to avoid the hazards and complications of jointing separate lengths at sea by means of rigid submarine joints. The discontinuity in torsional and longitudinal strains and bending, which such joints impose on the cable during laying, were regarded as highly undesirable. Consideration was given to the manufacture of cable by conventional methods in the longest lengths practicable, and their jointing in the factory by reconstituting the cable ('reconstituted joints') and armouring and serving overall into the continuous lengths required for laying. Such reconstituted joints are regions of possible weakness, both electrical and mechanical, since they are dependent to a large extent on the human element. It was therefore decided that, though such joints must be fully developed, their use involved uncertainty which could only be justified under conditions of emergency. The decision was accordingly taken to manufacture the cables in continuous lengths, and to develop and install the special plant for manufacture and handling necessary to achieve this.

This concept of continuously manufactured lengths had received much support from the evidence and experience obtained from previous work carried out in conjunction with the Central Electricity Authority on the cross-Channel cable project.[1,2]

The decision to adopt the single-core gas-filled type of cable using pre-impregnated paper insulation[3] followed logically, this being the only practicable type which lent itself to continuous manufacture and which could be developed for the duty required.

(6.2) Cable Design

The cable was required to operate at 138 kV 3-phase 60 c/s, with a rating of 120 MVA. The maximum external sea-water pressure would be 265 lb/in^2, and in order to prevent extensive ingress of water into the cable in the event of damage, it was designed to operate at 300 lb/in^2 nitrogen gas pressure.

The design aspects to which development effort was particularly directed were as follows:

Suitability for operation at a depth of 100 fathoms.
Satisfactory gas transmission.
Mechanical properties to withstand the arduous conditions involved in handling and laying.
Adequate protection against corrosion, and resistance to fatigue.

Laboratory tests were not of themselves considered adequate, and two extensive sea trials were carried out at Loch Fyne, using the Post Office cable-ship *Alert*. These trials, which included laying, grappling and recovering the cable, were carried out in depths of approximately 100 fathoms.

The constructional details of the cable design which finally energed from the development stage and the two sea trials are briefly summarized in Table 3 and illustrated in Fig. 5. The cable comprises the following:

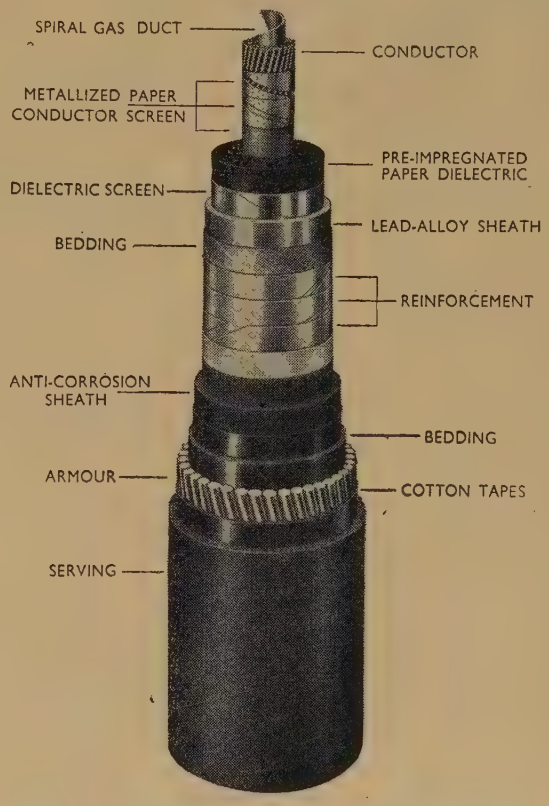


Fig. 5.—Construction of 138 kV 0·35 in^2 single-core gas-filled cable.

A circular screened copper conductor of area 0·35 in^2 (445 000 c.m.) with central gas duct.
Pre-impregnated paper insulation, copper-tape screened.
Lead-alloy sheath reinforced with bronze tapes.
Anti-corrosion sheath, galvanized-steel-wire armour and serving.

Features of particular interest associated with the design are as follows:

Conductor and Insulation.—The cable was designed on the basis that all coiling would be in a clockwise direction. The outer layer of conductor wires was stranded right-hand, so as to contract in diameter under coiling, thus avoiding damage to the dielectric papers. The central duct was designed to provide the gas-flow channel; the clearance under the lead sheath—which is standard practice in the case of land cables—is inadmissible, since

Table 3

CABLE CONSTRUCTION

Component	Radial thickness	Cumulative diameter
	in	in
Spiral steel duct (internal dia. 0·47 in)	0·030	0·530
Copper conductor 42/·105 in 0·35 in² (445 000 c.m.)	0·210	0·950
Metallized paper conductor screen	0·010 5	0·971
Pre-impregnated paper dielectric	0·575	2·121
Copper-tape dielectric screen	0·003	2·129
Alloy E (B.S. 801: 1953) sheath	0·150	2·429
Reinforcement bedding	0·012	2·453
Tinned 1% tin-bronze reinforcing tapes (3 × 0·009 in)	0·027	2·507
Compounded cotton tapes	0·027	2·561
Vulcanized-rubber anti-corrosion sheath	0·120	2·801
Armour bedding	0·118	3·037
Single-wire armour, galvanized steel, 25–32 tons/in² tensile	0·232	3·501
Bituminized cotton tape	0·050	3·601
Jute serving	0·180	3·961

at depths in excess of 40 fathoms the sheath would tend to deform under the external water pressure. The insulating papers were applied with graded thicknesses and tensions in four reversed-lay sections.

Gas Retaining Sheath and Reinforcement.—Alloy E was adopted for the sheathing, after careful study of the properties of alternative alloys, and having particular regard to the necessity for high fatigue resistance. The diametral strain, imposed on the sheath in the course of transferring its hoop stress to the reinforcement in service, was determined as 1·2% maximum, while the ultimate diametral strain of the sheath itself is 4·0% minimum at rates of strain down to 10^{-4}% per hour. The sheath was extruded on to the insulation with a shrinkage of 30–40 mils diametral, to obviate distortion under external water pressure while the cable was being laid without internal gas pressure.

The reinforcing tapes were applied with a right-hand lay (i.e. opposite-hand to the armour) in three layers, so that, under the tensile loads experienced during laying, their circumferential torque was opposed to and practically equal to that of the armour. Under coiling conditions, the tensile strain imposed on the tapes was within their elastic limit. This substantially reduced any tendency of the cable to throw loops on the sea bottom under certain conditions of laying and recovery.[4]

Anti-Corrosion Sheath.—This consists of a homogeneous and fully vulcanized sheathing of natural rubber compound, and performs the highly important function of protecting the reinforced gas-retaining sheath against corrosion. It provides adequate insulation against the transient voltages (approximately 10 kV) which may be induced between the armour and the gas-retaining sheath in service.

Steel-Wire Armour.—This consists of galvanized steel wires applied with a left-hand lay to avoid locking during the clockwise coiling. The optimum lay was determined, by experiment, as 12 times the pitch-circle diameter, to provide the desired coiling and handling properties consistent with minimum torsion under the tensile loads of laying. Much work was done on various aluminium alloys with the object of reducing the electrical losses in the armour. Although their normal electrical and mechanical properties were satisfactory, it was found that, under water, the abrasion resistance was much inferior to that of steel, and for this reason the use of aluminium wire armour was rejected and galvanized steel was adopted.

Technical Characteristics, Tests, etc.—The technical characteristics of the cable, together with brief details of typical performance tests, are given in the Appendix (Section 16).

(7) DESIGN OF TERMINAL EQUIPMENT AND JOINTS

The use of the hollow cable conductor as a gas duct made it necessary to modify the conventional sealing-end design which is used for land cables, as the feed pipe was at line potential. The general arrangement is shown in Fig. 6, from which it may be seen that a double porcelain arrangement was employed.

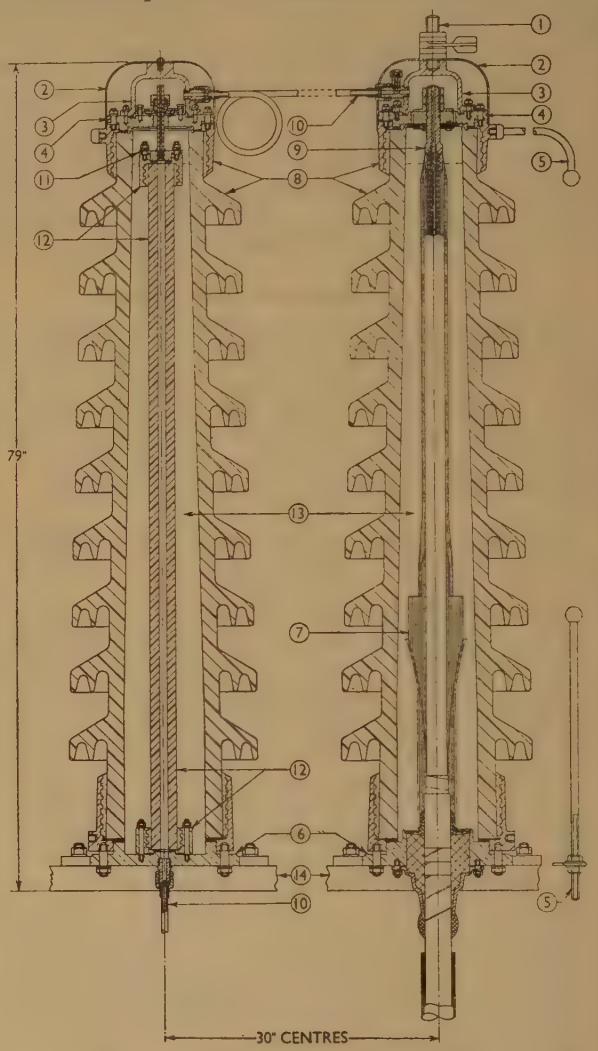


Fig. 6.—Cable-termination sealing end.

1. Outgoing connecting stalk.
2. Spun copper corona shield.
3. Gas-entry dome.
4. Intermediate sealing plate.
5. Arcing horns (adjustable lower horn).
6. Base plate.
7. Impregnated paper roll forming stress relief cone.
8. Porcelain insulator with cemented-on end rings.
9. Copper connector.
10. Gas-feed pipe.
11. Top sealing plate on gas-feed porcelain.
12. Gas-feed porcelain with cemented-on end rings.
13. Filling compound.
14. Hardwood mounting batten.

For maintaining gas pressure in the cables, gas-feed equipment was installed at each terminal site. The gas cylinders were connected to the sealing ends through the usual arrangement of reducing valves and pressure gauges, the latter being provided with electrical contacts to give warning of falling pressure, either from cylinders or cables, in the event of a leakage.

A novel system was incorporated, whereby the maximum rate of gas flow would be automatically controlled to a value just sufficient to maintain the pressure necessary to prevent the ingress of water in the event of severe damage to a cable, such as, for example, a complete severance. Provision was made for gas-leak location panels at each terminal site.

As explained earlier, it was intended that the cables should be manufactured and laid without joints. It was appreciated that this ideal might prove to be unattainable, and so joints of two types were developed. The first was a reconstituted joint of the same dimensions and flexibility as the cable itself; this was intended for use in the factory if an accident should occur at any stage up to and including lead sheathing. The second type of joint was a rigid submarine repair joint for use if a cable should need repair after laying. The joints are described below, and, as it transpired, both were needed in the project.

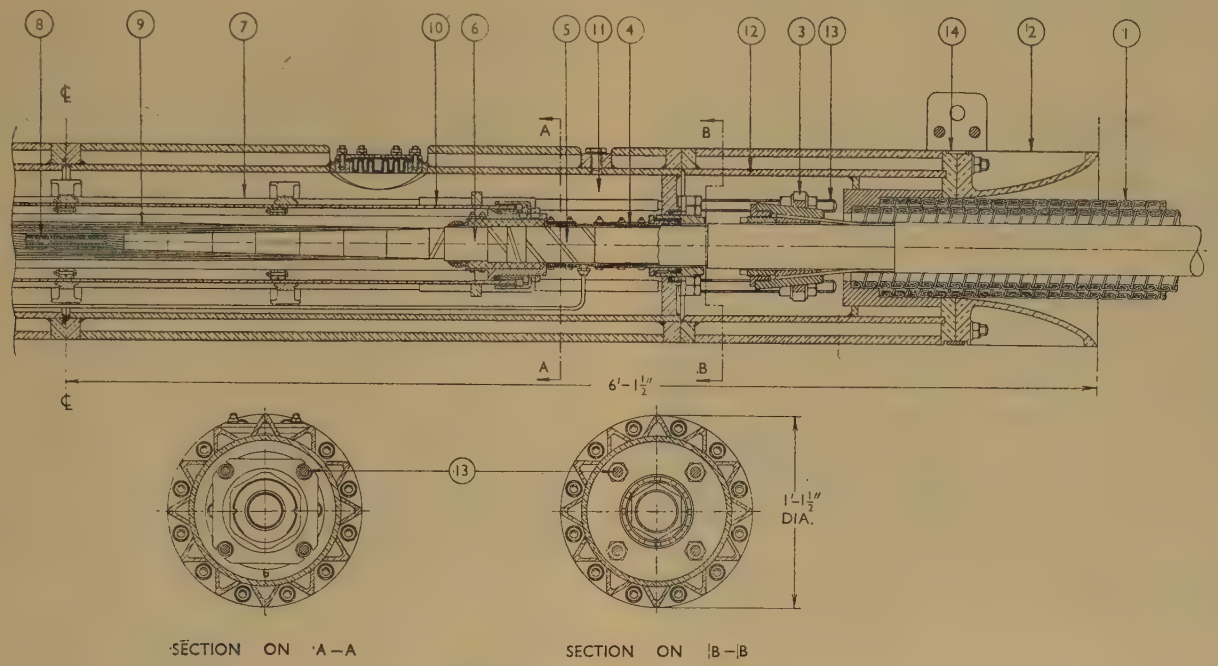


Fig. 7.—Submarine repair joint.

1. Flexible metallic bending restrictor.
2. Aluminium-alloy flare.
3. Steel armour clamp.
4. Rubber sheath.
5. Cable reinforcing tapes.
6. Lead sheath of cable.
7. Split copper sleeve.
8. Copper mandrel and ferrule.
9. Paper-tape joint insulation.
10. Monel pressure sleeve.
11. Bituminous compound.
12. Aluminium-alloy outer shell.
13. Steel tie rods.
14. Steel lifting assembly.

Overall length: 12 ft 3 in.
Overall diameter: 1 ft 1½ in.
Gross weight: 660 lb.

In the flexible, reconstituted joint, the conductor was joined by first removing a portion of the central duct spiral and replacing it by a smaller spiral with a single layer of copper wires over it. The two layers of conductor wires were relaid over the insert, and each pair of corresponding wires was joined by butt-brazing with silver solder. Joints in adjacent wires were made at alternate ends, whilst the joints in the two layers of the strands were staggered. The extra wires on the spiral insert were for the purpose of compensating for the slightly increased electrical resistance of the brazed joints and for the rather higher thermal resistance of the subsequently hand-applied dielectric. After the conductor had been joined, the conductor screening, the dielectric and the core screening were applied by hand-lapping, the original cable dielectric having been tapered in the usual manner. An oversize lead-alloy pipe of the same composition as the cable sheath, which had been threaded over one end of the cable before jointing commenced, was then drawn over the joint and swaged down to the cable-sheath diameter by means of a special tool. The ends of the pipe were then joined to the cable sheath by lead-burning. The slight protuberance of the lead burns was gradually sloped down to the cable sheath by varnished-silk-tape lappings; this was necessary so that the reinforcing tapes would be applied over a smooth profile. Selected cable jointers and skilled lead burners were trained in the art of making of such joints until an electrical and mechanical performance practically as good as that of the cable itself was consistently attained.

The submarine repair joint is illustrated in Fig. 7. The electrical design was virtually the same as that proved by land usage, but it was necessary, in addition, to make the joint strong enough to withstand handling under the arduous conditions of submarine laying. Moreover, materials had to be carefully selected to avoid corrosion.

Flares and flexible metallic tubing were used at the ends of the joint to prevent undue bending of the cable where it emerged from the joint carcase. An essential feature of submarine cable joints is that the time required for making them must be reduced to the absolute minimum because of the weather hazard. In this case, the design and the jointing procedure were refined step by step until a time of 24 hours from start to finish was achieved.

(8) CABLE MANUFACTURE

(8.1) Details of Plant

While development work on the cable leading up to the final design and manufacturing specifications was proceeding, design and installation of the manufacturing plant, coiling sheds, and

transport and loading gantries were going on. In designing the plant and its layout for the manufacture of the continuous lengths of cable required, particular consideration had to be given to the method of storage of the long lengths of cable in between the various stages of manufacture, and to the best means of enabling the insulating and lead-sheathing operations to be carried out in tandem. It was clearly not satisfactory to leave the insulated and screened cable exposed to the atmosphere for many weeks.

The plant was designed for the following sequence of operations, which are illustrated diagrammatically in Fig. 8:

(*a*) Stranding of conductor on to No. 1 turntable.
(*b*) Strand from No. 1 turntable insulated and lead-sheathed in tandem, using an accumulator system, the lead-sheathed cable being taken up and stored on No. 2 turntable.
(*c*) Lead-sheathed cable from No. 2 turntable reinforced, vulcanized-rubber sheathed, armoured and served; and the finished cable conveyed by gantry and coiled down in the coiling shed.
(*d*) Cable transported from the coiling shed by overhead gantry and loaded into the tanks of the cable ship for shipment to site and subsequent laying.

Much of the plant and equipment employed follows existing design practice, and only those items which have been specially developed for the project are briefly described.

Stranding Machine.—This is a standard floating-bobbin 2-carriage machine. Novel arrangements had to be made to take the conductor away from the capstan and feed it on to No. 1 turntable. A capstan system of haul-off will not operate with zero output tension owing to slipping, and since the strand has to be fed from the strander on to the turntable at zero or very low tension, a small caterpillar haul-off is employed on the output side of the capstan to maintain a small output tension. This caterpillar haul-off is fitted with Neoprene pads driven through a fluid drive and is mounted over the capstan so that the conductor may be fed on to the No. 1 turntable from above.

Storage of Conductor (*No. 1 Turntable*).—The normal method of storing a long length of cable is to coil it on the ground, but this twists the cable through 360° for every turn of the coil. It was decided that twisting of the cable should be reduced to a minimum at every stage of manufacture except in its finished state, and that large power-driven horizontal turntables should be installed for coil-storing the cable between operations.

To receive and coil-store the conductor, such a turntable, 24 ft in diameter, was installed between the stranding and insulating machines designed to accommodate up to 25 miles of conductor of 1·0 in diameter. It is driven through an electronically-controlled magnetic coupling giving constant speed at any load. Owing to the difference in speeds between coiling down from the stranding machine and paying-off into the insulating machine, the drive was fitted with two gears and a 2-speed motor.

Insulating Machine.—This is a rotating-cage-type lapping machine of special design, having four cages each capable of applying 50 paper tapes. The heads are of a constant-tension type regardless of paper-pad diameter, and they give true tangential and precision application; a separate head applies the copper screening tape. As the paper applied to the cable is pre-dried and impregnated, special measures are taken to eliminate contamination and to control the moisture pick-up during any enforced long stoppages of the machine. The insulating machine is therefore fully enclosed with a drying plant to maintain the atmosphere within the enclosure at uniform humidity, which can be reduced to 3–4% in the event of a prolonged stoppage. Fitted into the sides of the enclosure are a number of cabinets for storing the pads of paper after cutting. These have doors on both sides so that the pads can be fed into them from outside, for use by the operators working inside the insulating machine enclosure. The humidity of these cabinets is maintained at 3–4%.

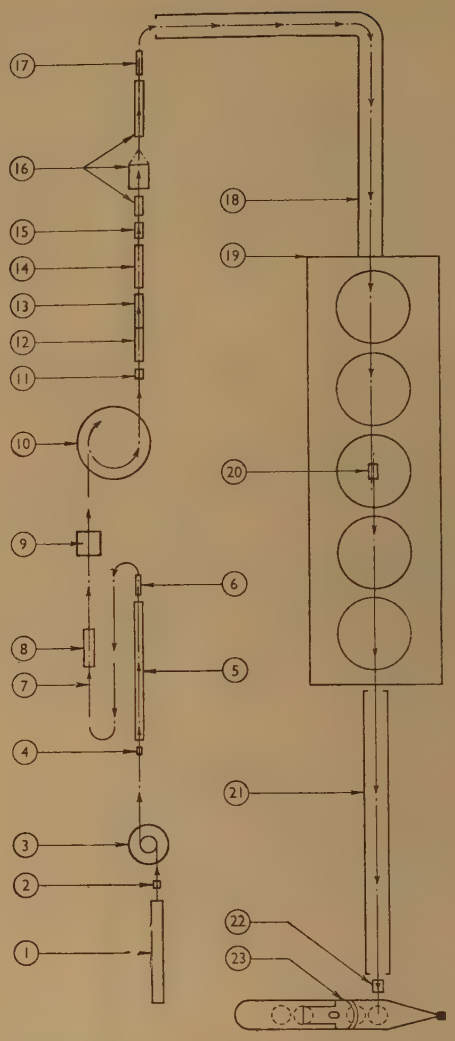


Fig. 8.—Diagram of plant for continuous manufacture of submarine power cables.

1. Stranding machine.
2. Caterpillar haul-off.
3. No. 1 turntable.
4. Caterpillar brake.
5. Insulating machine in enclosure.
6. Capstan haul-off.
7. Accumulator system.
8. Accumulator output capstan.
9. Lead press.
10. No. 2 turntable.
11. Caterpillar brake.
12. Reinforcing tape heads.
13. Rubber taping heads.
14. Vulcanizer.
15. Spark testers.
16. Jute bedding, wire armouring and serving machines.
17. Capstan haul-off.
18. Transport gantry.
19. Coiling shed.
20. Coiling winch.
21. Loading gantry.
22. Loading caterpillars.
23. Cable ship.

Accumulator between Insulating Machine and Lead Press.—As has been explained, it is necessary to run the insulating machine and lead press in tandem. It is, of course, impracticable to run the two machines synchronized together as this would necessitate stopping the insulating machine for several minutes whenever the lead press stopped for recharging, and, even more important, stopping the lead press whenever the insulating machine stopped for renewing papers. An accumulator system was designed and installed between the two machines to store 130 yd of cable, and

so enable the lead press to run for up to three hours at normal production speed with the insulating machine stopped; in an emergency the lead press could run for up to 20 hours should a breakdown of the insulating machine require it.

The accumulator consists of a large 8-fall pulley system mounted horizontally. One set of four pulleys is fixed and the other set of four is moving, being mounted on a trolley which runs on rails erected over the insulating machine; all the pulleys are of 15 ft diameter. A dead-weight system provides the necessary tensioning between the fixed and moving pulleys, the spans of cable between them being supported by a system of rollers. The system is shown in Fig. 9.

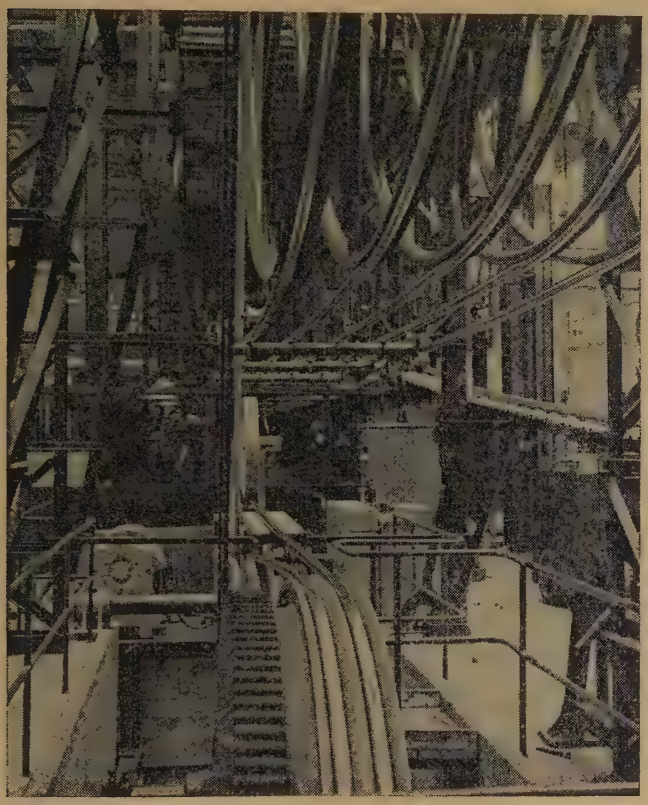


Fig. 9.—Accumulator for cable storage between insulating and lead-sheathing.

The cable is metered out of the accumulator system to the lead press by means of a double capstan passing the cable in a catenary on to a roller conveyer which leads the cable into the lead press. This capstan is driven by an electric motor through a variable-speed gear, control of which is by means of a roller resting on the cable in the catenary. When the lead press starts or speeds up, the cable lifts slightly, raising the roller, which, in turn, starts or speeds up the capstans; the arrangement thus automatically meters the cable from the accumulator into the lead press at low and uniform tension and at precisely the speeds required.

Lead Press.—In a programme of continuous manufacture such as this, it was essential to ensure that the plant would be capable of running continuously for periods of 500 hours or more. From this aspect, the most vulnerable unit was the lead press, where a stoppage of 10 min would so damage the cable as to necessitate cutting it out.

The press employed was a standard 1 000-ton upstroking type with jack-ram return, fitted with the Glover vacuum method of filling;[5] prolonged trials had confirmed that this was suitable for the continuous extrusion of Alloy E sheathing for periods of up to 30 days.

As a safeguard, the pumps and control console were duplicated, any necessary change-over being possible within a few minutes. The hydraulic pressure was limited to 200 lb/in^2 during a pre-set period following recharging, thus rendering it impossible, by starting extrusion prematurely, to cause a lead blow up the ram.

A special γ-ray photographic unit was developed for measuring regularly the sheath thickness at four positions at right angles to each other, without interruption of production. The γ-ray source is 2 curies of cobalt 60 in a heavily screened container with a remotely controlled shutter.

Storage of Lead-Sheathed Cable (*No. 2 Turntable*).—A second turntable, 47 ft in diameter, was installed between the lead press and reinforcing machine; this will accommodate up to 25 miles of cable measuring approximately 2·5 in in diameter. Fig. 10

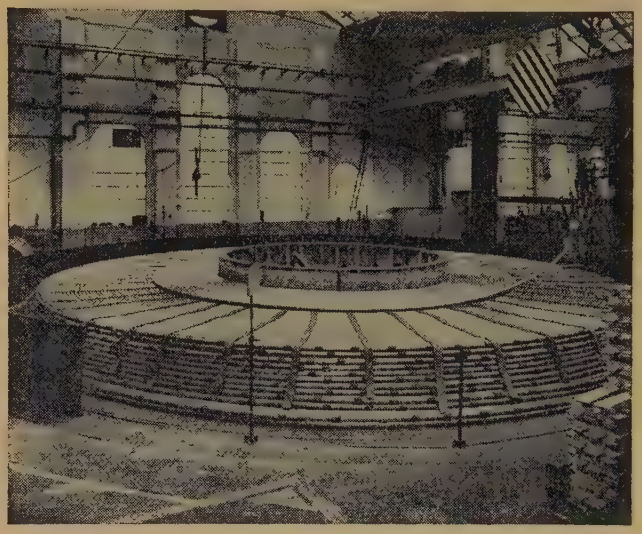


Fig. 10.—Storage turntable for lead-sheathed cable.

shows this turntable loaded with approximately 15 miles of lead-sheathed cable.

Reinforcing and Armouring.—The application of the reinforcing tapes, vulcanized-rubber anti-corrosive sheath, bedding, steel-wire armour and serving is carried out in tandem in machines driven from a common backshaft.

Reinforcing Machine.—This machine is used for applying the bedding, metal reinforcing tapes, cotton tapes and unvulcanized rubber, and the bituminized tapes applied very tightly over the rubber for the vulcanizing process. The heads were designed to give tangential lapping, and the metal tape heads were specially designed for the high tensions required.

Vulcanizer.—This consists of four compound pans, each about 9 ft long and containing heated bitumen compound. A rotating drum picks the compound up from the pan, and a steel blade deflects the compound from the drum and directs it on to the cable and onto a trough which ensures effective immersion. The blade deflecting the compound, and the trough, are moved into position when the machine starts, and out of position when it stops. This allows the vulcanizing process to be carried out only when the cable is moving; the standard time of immersion is fixed at 3 min.

Spark Testing.—To ensure complete integrity of the rubber sheath over the reinforcing tapes, it is tested at 20 kV by two spark testing units in tandem. These are interlocked with the main motor drive so that voltage is applied whenever the cable is moving, and the motor cannot run if the voltage to the spark

testers fails. In addition, a failure in the rubber sheath automatically stops the armouring-machine motor to allow inspection and repair to be made.

Armouring Machine.—This is a standard wire-armouring machine with jute heads and compound pans, and calls for no special comment.

Storage of Finished Cable.—The cable is transported from the capstan of the armouring machine by overhead gantry and coiled in the coiling shed (see Fig. 11). In order to coil the cable

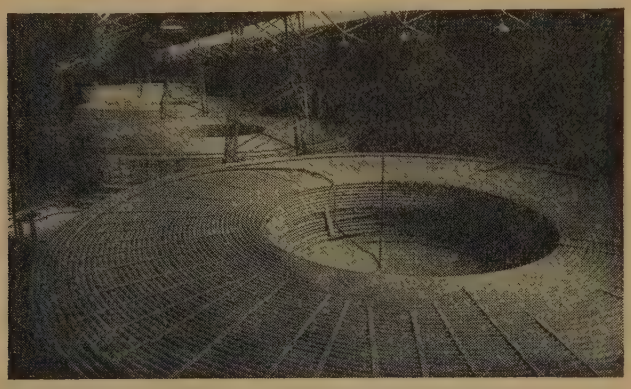


Fig. 11.—Coiling shed for storage of finished cable.

satisfactorily it is necessary to lower it from a height, and for this purpose a structure was erected comprising towers 45 ft high and 64 ft apart, with a bridging gantry between them. In the middle of the gantry a winch was installed, consisting of a 10½ ft diameter V-grooved wheel driven by an electric motor through fluid coupling. The starting and stopping of the winch is controlled by a loop of cable between the armouring-machine capstan and the start of the overhead gantry.

Loading into the Cable Ship.—To load the cable into the cable ship for shipment and laying, an overhead gantry was built from the coiling site to the dockside—a distance of 350 yd. To enable the cable to be loaded without the help of ship's gear (thus permitting the cable to be loaded into a freighter, if necessary), the gantry is terminated with two cantilevered extensions from the dockside tower over the hold of the ship. These cantilevers are pivoted at the gantry end, so that they can be rotated into correct position over the ship's hatches. Two caterpillar winches, each capable of exerting a pull of 4 tons, were installed on the dockside tower, and these are able to pull the cable from the coiling site and push it along the cantilever, from which it is lowered for coiling into the tanks of the cable ship. Fig. 12 shows the cable being loaded.



Fig. 12.—Loading cable into cable ship *Ocean Layer*.

(8.2) Quality Control

It was not practicable to test the cable under high-voltage a.c. conditions and full gas pressure, either during or on completion of manufacture, owing to the lengths involved, and to the fact that full gas pressure could not be applied without impairing the flexibility and handling properties of the cable before laying. It was therefore essential to take quite exceptional measures to control the uniformity and quality of the cable throughout all stages of manufacture; these are briefly summarized as follows:

(*a*) Prior to the commencement of manufacture, a very comprehensive system of specifications was prepared, covering all manufacturing materials, processes, and constructional details, and defining all testing methods. These specifications were rigidly adhered to throughout.

(*b*) Inspectors were carefully trained, and employed in shifts to provide continuous inspection of all aspects of manufacture, including control of materials. All inspection data were continuously charted and analysed statistically, so that corrective action could be taken in the initial stages in the event of any undesirable trends developing. The calibration of control instruments, gauges, etc., was frequently checked.

(*c*) Control tests on the pre-impregnated paper tapes prior to release for insulating were particularly comprehensive and stringent. It will be appreciated that this control, allied to the frequent checking of dimensions, tensions, etc., during insulating, ensured a high standard of quality and uniformity of the cable dielectric.

(*d*) At the start and finish of each length manufactured, lengths were cut off for complete testing and dissection to confirm compliance with the specifications. High-voltage d.c. tests and pneumatic tests at reduced gas pressure were carried out on the full lengths of cable in the lead-sheathed stage (No. 2 turntable) and the finished stage (coiling shed).

(8.3) General

After the plant had been installed, it was run in over a period of several weeks to eliminate initial difficulties, prove its performance, and train personnel. A trial length of cable was then manufactured and type-tested for full compliance with the specification before the plant was released for manufacture of cable for the project.

Manufacture began on the 1st April, 1955, and was completed on the 2nd March, 1956, the total length manufactured being 93·5 miles, comprising five lengths of approximately 3 miles and five of approximately 15·7 miles. Generally, manufacture proceeded very satisfactorily and as planned. There were, however, three mishaps—two due to plant, and one resulting from a total power failure—which necessitated the insertion of reconstituted joints.

(9) CABLE INSTALLATION

(9.1) Shipment

The cable was shipped to site and laid by the cable ship *Ocean Layer*.[6] This ship, of 4 600 tons displacement, had recently been completely fitted out with the most modern equipment and gear, and was, with her experienced captain and crew, to prove very suitable for the difficult installation operations.

Cable was loaded into the tanks of the *Ocean Layer* by coiling clockwise 'in and out' using a minimum of timber packing and an initial eye diameter of 13 ft. Loading was in continuous shifts at an average coiling speed of about 800 yd/h. Fully loaded with approximately 3 900 tons of cable, the *Ocean Layer* sailed for Vancouver and commenced laying operations there in Trincomali Channel on the 16th July, 1956.

(9.2) Landing Sites

Very detailed surveys were made of the waters and bottom conditions surrounding the landing points. Storm records over a period of years were examined, and it was established that the effect of wave action could be adequately restrained.

The three island terminals of the submarine cable routes are broadly similar, lying in small identations on rocky coastlines with sufficient depth for the cable ship to anchor about a quarter

of a mile off shore in 15–25 fathoms. The aspects of these inlets requires a somewhat dog-legged approach in all cases. After clearance and grading of these inlet sites, reinforced-concrete chases channelled to receive the cables were constructed from 5 ft below mean low water to the sealing-end sites some 50 ft above high-water mark and 300–450 ft distant. These were subsequently covered with concrete slabs. Additional protection for cables exposed at low tides was achieved by sleeving them with split articulated protectors of malleable cast-iron for a short distance out from the shore.

The mainland terminal at Tsawwassen Beach presented special problems, since it has a sea approach over about 1½ miles of sandy shallows, and on land a steep unstable bluff rising to a height of 180 ft. This bluff was cut to a depth of about 13 ft to reach a competent layer, and a concrete cable subway 450 ft long furnished with steps and cable racks was constructed leading to the sealing ends some 900 ft from the high-watermark. On completion, the cut was filled and replanted to reduce the risk of subsequent movement.

(9.3) Installation Methods

The greater part of each cable was laid from the cable ship using established methods. In order to maintain correct spacing of the cables under the somewhat difficult tidal conditions, it was decided to bring the ship to a pair of mooring buoys roughly parallel to the shore near the end of each crossing.

The use of continuous cable lengths precluded laying shore ends separately, and special attention was focused on the problems of getting the ends ashore. Scows, or barges with shallow draught and large flat decks, are used extensively in the Vancouver area, and it was decided to adapt these local craft for laying cables from ship to shore. For a starting end the required cable length would be coiled on to a scow adjacent to the ship at moorings and would be laid from the scow coil to the shore, where the balance would be pulled by the free end up the cable chase to the sealing ends using a power winch. After laying cable by ship to the finishing-end moorings, a bight would be pulled on to the scow deck and the remaining length transferred from the ship and coiled down before laying to the shore and pulling to the sealing ends.

Cable lengths to be transferred to the scow at the starting- and finishing-end moorings would be around 2000 ft in all cases, except at the mainland, where they would be a little over 9000 ft, their weights being approximately 16 and 72 long tons, respectively. It would be necessary to devise gear for carrying out this work and to have it fitted to suitable scows, bearing in mind that the twist introduced in coiling must be removed as the cable was laid. This would happen normally at a starting end when the scow leaves the ship paying out cable towards the shore, and also at a finishing end when the cable is pulled up the chase from the scow. However, pulling up the chase at a starting end and laying from ship to shore at a finishing end would introduce a double twist in the cable if paid out in the normal way. It was proposed to obviate this difficulty by paying out cable under the turns of the coil.

(9.3.1) Cable Scow.

The problems of handling shore ends were studied using models (see Fig. 13) and later a full-size dummy scow and ship on land (see Fig. 14). It was confirmed that 'double twist' could be avoided by paying out the cable from underneath the coil through a 'tunnel', and using a device called a 'spinner'. This is a semicircular frame with rollers suspended above the cable coil and free to rotate about a vertical axis. Cable is led over this, down round a set of rollers at deck level and under the coil through the 'tunnel' (see Fig. 15). This amounts to paying out



Fig. 13.—Use of models to determine installation methods.
Pulling bight of cable on to scow deck at finishing end.



Fig. 14.—Use of dummy scow and ship on land to determine installation methods.
Ship's bow sheave is in the foreground and cable-scow winch structure is in the background.

the coil from underneath, which is equivalent to turning it over. A clockwise coil becomes counterclockwise when viewed from underneath, and so the twist imposed by clockwise coiling viewed from above is removed by a counterclockwise movement as cable is virtually paid out from the bottom of the coil.[7]

As a result of this work, a cable scow was designed as shown diagrammatically in Fig. 16. A reversible power-driven cable sheave is mounted centrally 25 ft above the coiling area on deck on an A-frame structure. Intended primarily for coiling down cable transferred from the ship and for laying the long shore approach to the mainland, it is driven by hydraulic variable-speed gear also arranged for braking. A compression-type cable brake is fitted at the stern, so arranged that the cable can be lifted in and out and the brake can be removed bodily in sections to clear the deck for handling cable bights. A tunnel consisting of wooden ramps with a removable top plate is provided so that cable can be paid out under the remaining turns as previously described. At the stern, a rounded 'apron' is built up and extended round the sides merging into vertical curved plates so that cable can be pulled off in almost any direction without too sharp a bend. The scow is equipped with winches for warping and cable handling. V.H.F. radio and loud hailers, a compact galley and store below deck and a navigation platform at the top of the structure complete the equipment.

Final trials were carried out at sea near Falmouth in December, 1955, using the *Ocean Layer* and the scow equipment and ancillary

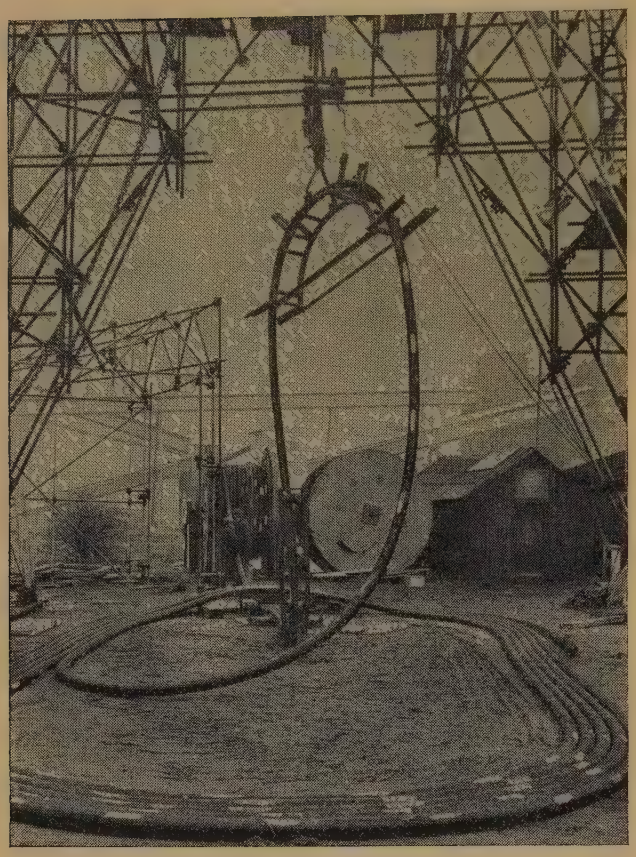


Fig. 15.—Tests on land.

Use of 'spinner' to obviate double twist.

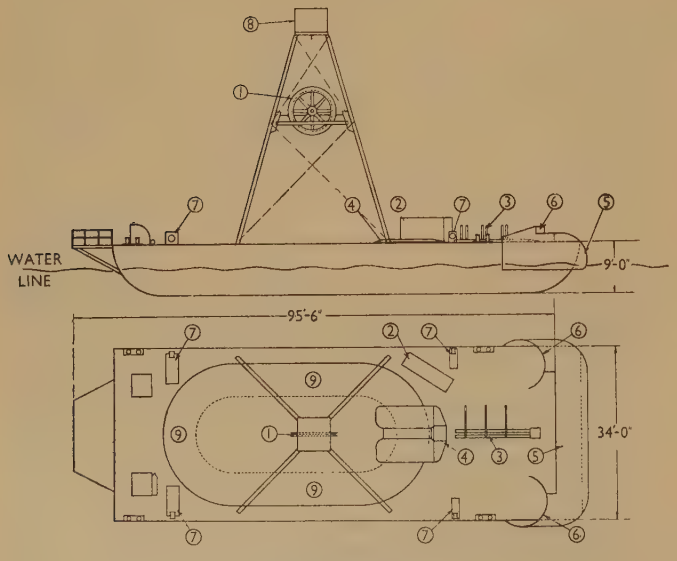


Fig. 16.—Cable scow.

1. Cable sheave.
2. Prime unit.
3. Cable brake.
4. Tunnel.
5. Stern apron.
6. Curved side plates.
7. Winches.
8. Navigation platform.
9. Cable coiling area.



Fig. 17.—Frame for lifting cable on and off scow sheave.

gear as described, mounted on a pontoon of suitable dimensions. These trials proved the equipment and methods as satisfactory for laying of the starting and finishing ends. In addition to the fixed equipment on the scow, a simple lifting frame of the required radius was used for getting the cable on and off the sheave wheel; this is illustrated in Fig. 17. A 'bight puller', consisting of a series of rollers on hinged frame sections with bending limited to a safe value, was also used for pulling the cable bight on to the deck of the scow at a finishing end (Fig. 13).

Two wooden scows, 90 ft long by 34 ft wide and about 230 registered tons, were fitted out and used to rehearse methods and train their crews prior to commencement of the actual installation.

(9.3.2) **Laying from Scow.**

For the laying and landing of the starting ends a scow was moored alongside the ship, and the required length of cable was transferred using the cable sheave. Except at the mainland, the scow was then warped to the shore, paying out cable from the coil, over the cable sheave, through the brake and over the apron. At the mainland, however, the scow was towed for the first mile by shallow-draft tugs to dolphins about half a mile out, where ropes were picked up for warping to the shore. The tension in the cable was maintained by adjusting the speed of the sheave to maintain an angle of cable entry into the water of approximately 60° to the horizontal. When the scow arrived at its prepared position close inshore, it was moored, the cable was removed from the power sheave, and the spinner was rigged for the free end to be pulled away through the tunnel and over the apron to the terminal position. During this operation the tension on the cable on the sea side was taken by a stopper and rope attached to the scow. When the final bight of cable approached the apron, a further stopper was fitted on the land side so that all tension was removed from the bight. This was lowered over the apron by pulling the cable slowly ashore, at the same time releasing the tension on the two stoppers until the bight was pulled out.

At the finishing end the cable ship was moored to the buoys and the scow was moored in line ahead. A stopper was fixed to the cable and the weight of the suspended cable was transferred to the scow. Between the cable stopper and the bow sheave the bight puller was fitted over the cable. A bight was then pulled out over practically the whole length of the deck of the scow and the seaward cable was placed in the tunnel and the cable brake. The cable from the ship was lifted on to the scow sheave and the shore length was paid out from the ship and coiled down on the



Fig. 18.—Scow laying cable to shore at finishing end.

deck of the scow. By paying out cable to the shore from underneath the coil, using the spinner and tunnel previously described, it was laid without residual twist (see Fig. 18). When the scow had been warped into position at the landing site the cable was removed from the spinner and the free end was passed over the power sheave and pulled ashore. The final bight of cable was handled in the same manner as a starting end.

As soon as possible after laying, the shore ends were inspected by a diver, and subsequently articulated split cable protectors of malleable cast iron were fitted for a suitable distance from the foot of each chase. The cables were then buried on the foreshore by digging or water-jetting.

(9.3.3) **Laying from Ship.**

To avoid the risk of throwing loops or kinks in the cable, it was decided that it should be laid with a small residual tension on the sea bottom. A theory developed and checked during sea trials, which will be published later, was used to calculate the position of the 'touch down' point of the cable in relation to the position of the echo-sounder in the ship, the shape of the suspended cable, and also the tension required at the dynamometer on the deck to maintain the desired residual tension in the cable on the sea bottom.

The route profiles were plotted from hydrographic survey charts, and Tables were prepared for each cable for use by the laying controller, relating the dynamometer tension required at any instant to the water depth for the speed of laying. The only modification to this simple procedure was over a short distance on the downward side of the ridge off Galiano Island, when it happened that the ship was sailing in water of constant depth while the cable was still moving down the slope. In this region, the tension was controlled by reference to the length of cable paid out.

During the laying, comprehensive records were obtained, including depth, dynamometer tension, ship's course and speed, length of cable paid out, straight-line distance travelled by the ship (from a taut-wire apparatus) and angle between the cable and the water surface. From the data obtained it was confirmed that the cables had been laid closely in accordance with the planned requirements.

The speed of laying was limited to approximately 3 knots, which was considered the maximum at which the cable could safely be handled out of the hold and through the ship's machinery. During cable laying, the ship's navigating officers plotted her position at 4 min intervals using suitably positioned markers on land, and, in the Strait of Georgia, additional marker buoys with radar reflectors, previously laid by the ship. Laying proceeded without incident, except for one Trincomali Channel cable, when the ship was carried at right angles to the route by an unexpected tide rip near Parker Island, and the cable under heavy tension rose up over the bow-sheave flange and was bent to a radius of 15 in and visibly flattened. During d.c. testing after laying, it failed at this point and a submarine repair joint was inserted.

(10) FAULT LOCATION AND REPAIR

From numerous tests carried out, the pneumatic resistance of the cable had been shown to be very uniform, and accurate formulae were derived relating gas flow to the feed distance. The method developed for locating a submerged gas leak in a faulty cable consists of a loop test using a sound cable, applying a common pressure and measuring the relative gas flows, and then applying the formulae to calculate the fault position. Gas leaks in terminal equipment and cables above low-water mark can be located by methods used for land cables. The location of electrical faults, in general, follows conventional practice as for land cables.

Means had been developed for locating and cutting a faulty cable in the maximum depth of water prior to lifting for repair, since it would be impracticable to lift a bight of cable of this type in any appreciable depth without cutting it. The test methods already referred to, used in conjunction with the record of cables as laid, would give a preliminary location which might be confirmed using Leader gear[8] and by towing a submarine search coil to pick up a signal injected into the faulty cable. Grappling for the cable would then proceed in this area using a specially designed grapnel containing a search coil (to confirm that the correct cable was in the prong of the grapnel) and an explosive charge to be fired from the surface for cutting it.[9] The gas system previously described would limit ingress of water, and the two ends would be recovered separately for jointing-in the required new length of cable.

The fault previously referred to was located by the Hilborn[11] loop test, using other sound cables to complete the circuit, at a point some 600 yd from Parker Island in a depth of approximately 30 fathoms. The position of the fault made it possible to use a simplified repair procedure by cutting the cable near the foot of the chase and recovering it by scow to a point just beyond the fault position. A capacitor discharge was then used to give

precise audible location of the fault in the cable on the scow deck. It was confirmed that the cause was severe local crushing. The scow which recovered the faulty cable also carried a length of repair cable, a submarine repair joint, and all the necessary materials and staff. A four-point mooring was required owing to tidal conditions, and was provided by the ship's mooring buoys and two additional concrete clumps placed clear of the other cables. The faulty cable was cut out and the sound cable jointed to the repair length in approximately 24 hours by a team of jointers working continuously. The joint was lowered off the apron of the scow in line with the cable, the weight being taken by steel ropes to hand winches fixed on the scow. The repair cable was laid to the shore, the end being pulled to the terminal as for a starting end. It was later fully tested both pneumatically and electrically and made alive.

(11) GASSING, TESTING AND COMMISSIONING

The cables were charged with nitrogen to a pressure of approximately 10 lb/in^2 before they were loaded into the cable ship and were checked on arrival in Vancouver. Immediately after laying and before the terminations were made, the cables were tested at 276 kV d.c. for 15 min at atmospheric pressure. After the terminations had been made, the cables were gassed at a rate of 50 lb/in^2 per hour up to 350 lb/in^2 for a pneumatic over-pressure test. The cables were then retested at 276 kV d.c. for 15 min, and as soon as the circuit was available, line voltage was applied for a period of approximately 48 hours. The cables were brought down to their working gas pressure of 300 lb/in^2 and the pneumatic system was set to give the required warnings.

After all the cables had been subjected to these tests, the circuit was switched in on load without incident, and the spare cables were maintained at line voltage.

(12) OPERATING RECORD TO DATE

Since the submarine cable interconnection first went into official service on the 25th September, 1956, it has been in continuous operation except for a few short outages for testing and overhead-line maintenance. No operating problems have been experienced, except for a minor leak of the gas-pressure supply system at Tsawwassen Beach, which was remedied, and the interconnection has functioned satisfactorily and efficiently. Table 4 gives operating data for the interconnection to the end of 1956.

Table 4

BRITISH COLUMBIA–VANCOUVER ISLAND CABLE INTERCONNECTION

Monthly Energy and One-Hour Peak

Month in 1956	Energy	One-hour peak
	kWh	kW
September ..	15 931 300	48 200
October ..	18 144 000	50 200
November ..	3 276 000	17 400
December ..	7 735 000	41 800

Recently, during unexpected outages of some hydro-electric plants on Vancouver Island, the submarine interconnection was called upon to provide additional peaking power from the mainland system. Thus, at a very early date, the cable has already accomplished its dual purpose of providing a standby supply as well as a normal source of power for Vancouver Island.

(13) CONCLUSIONS

While the intention of the paper is descriptive rather than analytical, the authors believe that the following conclusions may be drawn from the project:

(*a*) The submarine interconnection between the lower mainland and Vancouver Island systems of the British Columbia Electric Company has proved to be a practical and economic engineering proposition.

(*b*) The design, manufacture and installation of long continuous lengths of submarine power cables have been successfully achieved in practice.

(*c*) It is believed that the engineering advances, which have been described, now make practicable and economic the transmission of large amounts of electrical energy under water by means of submarine cables. Many of the engineering difficulties in the field of underwater power transmission have been overcome and it is likely that important applications will now develop. It is probable that progress will be further stimulated by the advances made in the field of high-voltage d.c. transmission.[10]

(14) ACKNOWLEDGMENTS

The authors' thanks are due to the directors of the British Columbia Electric Co., Ltd., British Insulated Callender's Cables Ltd., British Insulated Callender's (Submarine Cables) Ltd., and British Insulated Callender's Construction Co., Ltd., for permission to publish the paper.

The submarine cable interconnection described in the paper has been brought to a successful conclusion by the combined efforts of many individuals at all levels in the above-mentioned companies and other manufacturers and contractors, and with the help of the personnel of the cable ship and other vessels; of oceanographers, surveyors and consulting engineers; and of Canadian, British and American authorities. The authors gratefully acknowledge the part played by all these individuals.

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(16) APPENDIX

Technical Data of Cable and Typical Performance Characteristics

(16.1) Technical Data

The more important technical details relating to the cable are as follows:

Conductor area	0·35 in² (445 000 c.m.)
D.C. resistance of conductor at 20° C	0·070 5 ohm per 1 000 yd
D.C. resistance of sheath at 20° C	0·284 ohm per 1 000 yd
D.C. resistance of reinforcement at 20° C	0·471 ohm per 1 000 yd
D.C. resistance of armour wires at 20° C	0·119 ohm per 1 000 yd
Weight in air	52·3 lb/yd
Weight in sea	37·5 lb/yd
Minimum permissible bending diameter	6 ft
Minimum permissible coiling eye diameter	10 ft
Maximum permissible twist	36° per yard
Maximum permissible tensile load	6 tons
Tensile modulus	0·017 % per ton
Torsional modulus	0·3° per yard per ton tension
Apparent 60 c/s resistance of cable at operating temperature	
500 yd spacing	0·153 ohm per 1 000 yd
250 yd spacing	0·153 ohm per 1 000 yd
Apparent 60 c/s reactance of cable at operating temperature	
500 yd spacing	0·097 2 ohm per 1 000 yd
250 yd spacing	0·099 4 ohm per 1 000 yd
Capacitance	0·193 μF per 1 000 yd
Working voltage to earth	80 kV
Electrical stress at conductor	85 kV/cm
Electrical stress at screen	39·5 kV/cm
Dielectric loss angle (at working voltage 60 c/s)	
At 20° C	0·003 9
At 50° C	0·003 3
At 85° C	0·003 0
Permittivity	2·9
Charging current, 60 c/s per cable	5·8 amp per 1 000 yd
Current rating	500 amp
Losses per 3-phase circuit	
Conductor	60·5 kW per 1 000 yd
Dielectric	3·5 kW per 1 000 yd
Sheath reinforcement and armour	55·0 kW per 1 000 yd
Total	119 kW per 1 000 yd
Maximum conductor operating temperatures	
In sea	50° C
On land	70° C
Maximum sheath temperatures	
In sea	30° C
On land	47° C
Soil resistivity	
On sea bed	30 thermal ohm-cm
On land	120 thermal ohm-cm
Average ambient temperature	
In sea	15° C
On land	15° C
Cable spacing	
Strait of Georgia	500 yd
Trincomali Channel	250 yd

(16.2) Typical Performance Characteristics

The following are typical of the extensive performance tests carried out on the cable:

Electrical Stability.—Fifty heating cycles to 90° C, conductor temperature at 1·5 times the working voltage with 200 lb/in² gas pressure. The cable was then maintained at 90° C and 1·5 times the working voltage until thermal stability was proved.

Surge Withstand.—675 kV (peak) for ten positive and ten negative impulses, with the conductor at 85° C and the cable under 200 lb/in² gas pressure.

Bending Test.—The cable, at atmospheric pressure, was bent three times each way in reverse directions around a drum of diameter 20 times that of the cable sheath (i.e. 49 in), followed by a withstand test of 300 kV d.c. for 15 min.

Coiling Test.—The cable, at atmospheric pressure, was coiled down three times to a coiling diameter of 10 ft, followed by a withstand test of 300 kV d.c. for 15 min.

Bending Test under Tension.—The cable, at atmospheric pressure, was subjected to six bends around a 7 ft 6 in diameter sheave under a 6-ton tensile load, followed by a withstand test of 300 kV d.c.

External Water-Pressure Withstand.—The cable, at atmospheric pressure, was subjected to 400 lb/in² external water pressure for 14 days, with no distortion.

Internal Gas-Pressure Withstand.—The cable was subjected to internal gas pressure of 600 lb/in² for 14 days.

(16.3) Tests on Accessories

The reconstituted cable joint was subjected to the same performance tests as for the cable, except that the surge-withstand voltage was 600 kV.

The submarine repair joint was tested, as for the cable, for electrical stability, surge-withstand, external water-pressure withstand and internal gas-pressure withstand; in addition, tests were carried out to prove the performance under tensile loading and handling conditions.

The sealing ends were tested as for the cable for electrical stability, surge-withstand, and internal gas-pressure withstand.

DISCUSSION BEFORE THE INSTITUTION, 25TH APRIL, 1957

Sir Josiah Eccles: The project is surely a landmark, or at any rate a milestone, in the development of submarine power transmission. It is true that 132 kV submarine cables have been in operation between Sweden and Denmark since 1951, but the distance is only about 3½ miles. Since 1955, a d.c. cable of about 62 miles has been in operation between Sweden and the island of Gotland, but the voltage was only 100 kV d.c. So I think that this particular installation is a record for high voltage and distance up to the present.

I suppose that one of the reasons why it was possible to carry out the project at all was the fact that the British Columbia Electric Company system extended from the mainland to Vancouver Island, thus joining two parts of an existing co-ordinated system. I wonder whether, if the whole of Vancouver Island had been under the control of one supply company, it would have been necessary to do this so soon. Furthermore, if the whole of Vancouver Island had been under co-ordinated control, would it have been possible to move the site of the crossing substantially north?

In Fig. 1 Station 12 is not yet in commission. It will have an output capacity of about 250 MW and is the largest station contemplated in the system. I do not know the oceanography of the Straits, but if it had been equally feasible to cross higher up, say at Sechelt, it might have been easier to get output from the largest station straight across to the Island instead of going in through the rather complicated network which is primarily for supplying the city on the mainland.

This may be nonsense, because, from a small-scale map at 6 000 miles distance, it is unfair to make a comment of this kind, but it is the sort of thing that one would consider in a first appreciation of a situation, and it would hinge on the fact that there was a co-ordinated system of electricity supply over a wider area than there is at present. That is important in these very highly capitalized industries. In order to get the

maximum economy in capital expenditure, the electricity supply has to be integrated over a considerable area. I should like the authors to comment on that point.

The other half of the paper is devoted to the engineering problems of manufacture, and I commend it to younger engineers, in whatever branch they are engaged, as an example of the kind of thinking and detailed consideration that have to be applied at every stage of a large project if it is to be successful.

The cable manufacturers planned for the best but prepared for the worst. They actually designed and tested two kinds of joint, one for a fixed submarine installation and the other a flexible arrangement for emergency use in manufacture. In the event both were needed. That is the kind of foresight which one is pleased to pay tribute to. Furthermore the detailed arrangements described in the paper, and particularly those shown in the film, merit the consideration of young engineers who are very good in their theory but are not all fully fortified by experience.

In Section 3 it is stated that the transmission cost for this venture is 0·1 cent per kWh for a given load factor which is specified in the paper. The cost of electricity generated on the Island is also given. Then there is a general statement that the total cost of power delivered through this cable to the switching station on Vancouver Island will be lower than the cost with the local steam plant. Could the authors state how much lower, and what does that figure include? Does the cost calculation include the capacity of the existing network which is absorbed in this transmission scheme, or is it just the bare cost of the generation of electricity on the mainland and the cost of the transmission through the apparatus which is described in the paper?

There is an even more detailed point in Section 6.2 in connection with the gas-retaining sheath and reinforcement. The following statement is made:

> The diametral strain, imposed on the sheath in the course of transferring its hoop stress to the reinforcement in service, was determined as 1·2% maximum, while the ultimate diametral strain of the sheath itself is 4·0% minimum at rates of strain down to 10^{-4}% per hour.

Could the authors expand this? I would like more information on the significance of these percentages and to what they are related.

The paper is most stimulating and encouraging, particularly when one thinks of our own commitments on the cross-Channel cable between England and France. We have decided that this should be a 200 kV d.c. system with a conductor return, not an earth or sea return.

The arguments for and against the d.c. system are many. I will not attempt to enumerate them because in future, we may present a paper on the subject. But clearly if there are two different systems, such as the Continental and the British system, and an attempt is made to 'tie' them together by an a.c. connection of moderate capacity, there are difficulties. First, the power transmission is entirely dependent on fine control of the frequencies of the two systems. D.C. transmission gives an extra degree of freedom in that sense, but reactive power cannot be transmitted, and so each system has to be capable of supplying its own magnetizing current. That is not difficult where each system has a large capacity of generating plant associated with it. But if we have one-way traffic from a large system to one that has not much generating capacity of its own, it may be necessary to provide synchronous condensers or other means whereby the reactive power is generated at the receiving end.

Sir John Dean: I was very fortunate in being able to witness the first trials of the laying of the cables and the actual first layings themselves. I can substantiate everything Sir Josiah Eccles has said about the wonderful planning, the engineering and the excellence of the design of these cables. I have had some submarine-cable experience, and it is quite a problem designing a submarine cable, even of a normal size, which, after coiling and laying, will not have a tendency to kink or birdcage. With a cable of this size it is even more difficult.

After several trial runs, which enabled the captain of the ship to feel his tides and the seamen and cable hands to plan their work in an orderly manner, everything went splendidly. The *Ocean Layer* is fitted with what is known as a Pleuger rudder. This is interesting in that it has a 400 h.p. submerged motor and a propeller in it. It enables the cable ship to be manoeuvrable at very low speeds, which, with a normal rudder, would be impossible without tugs. The Pleuger rudder is therefore extremely valuable in cable laying, particularly for a job of this nature with such a heavy cable and in waters with such very difficult tides and eddies.

Monsieur R. Tellier (*France*): The honour which devolves upon me of representing Électricité de France is tinged to-day with profound sadness, as my presence is due to the premature passing of Maurice Laborde,* whose authoritative voice will be but imperfectly replaced by that of a modest collaborator.

The originality of the specification of the cables used for the remarkable installation under consideration lies, in particular, in a central duct for the passage of the gas, thus permitting the cable to withstand the high external pressure of the water.

However, the relatively large diameter of the duct may cause an important water ingress when the cable is severed due to external damage or intentionally for recovery, since, above a certain diameter, say 5 or 6 mm, an immersed pipe supplied with gas may give passage to water flowing in the opposite direction. To avoid this, it has been proposed in France to partition the gas duct into small elementary ducts, for instance, using a conductor made of hollow wires.

I have been very interested in the new method developed for the laying of terminal cable lengths. The spinner and tunnel method may be particularly helpful when the cable ship cannot stay close to the coast. However, the use of this special method can be questioned in the case of the installation of a single cable, such as the projected cross-Channel interconnection, since it is rather costly and requires very favourable sea conditions. It may be preferable to make a joint at sea, so as to divide the laying operation into successive independent stages.

It is important to note that the authors confirm that the cable was laid without any slack. The lifting of a power cable is thus practically impossible without cutting it on the sea-bed, and the device mentioned will prove very useful.

I think that the accurate location of gas leaks on a long submarine cable is difficult and may entail the lifting of a long cable length. Electrical fault location is easier, but few methods are available for a single cable. It is interesting to mention that in France the capacitor discharge method has been tested successfully several times, the discharge being detected from the surface of the sea.

For repairs made at sea the authors' opinion would be valuable as to the conditions for the making and immersion of a closing repair joint to avoid kinking the cable on the sea-bed. A possible method consists in having the spare cable length laid in the shape of a loop, to permit the final joint to be made between two parallel cable ends coming from the same direction.

Mr. D. P. Sayers: Section 2 of the paper gives a very good case for power transmission across the Straits, and I wonder why the project was not undertaken sooner. Was it due to technical doubts or legal difficulties?

No information is given about the actual capital costs of this project, but economically the 20-mile submarine cable is probably

* M. Laborde, Deputy Oversea Representative of the Council for France, died on the 12th March, 1957.

equivalent to about 200 miles of overhead line for which a high-voltage d.c. system might be more economic than an a.c. system. Was a high-voltage d.c. system seriously considered?

The longest cable link in the Georgia Strait crossing is 14·5 miles, with a maximum depth of 100 fathoms, compared with 25 miles and 30 fathoms in the English Channel crossing. From the authors' experience, which is the greater problem—distance or depth? With reference to Section 5, I do not understand the logic, or the economics, of installing five cables initially. Three cables and one spare would be suitable for the first circuit, leaving three cables to be laid when the second circuit was needed.

With regard to accessories, the second bushing (Fig. 6) seems an extravagant method of providing an insulated path for the gas feed.

The straight joint (Fig. 7) involves the making of three separate sleeves and much precision work by the jointer, and I think further efforts should be made to develop a simpler type of joint.

Section 8 rightly stresses the need for rigid quality control at every stage in the manufacturing process. One would expect that special steps would be needed to combat the inevitable boredom of operatives engaged on the same product and job hour after hour for the whole of the year during which this cable was in course of manufacture.

With regard to installation technique, the use of a scow with the special 'spinner' device seems the answer to the problem of working in shallow water such as that which exists for some three miles off the French coast near Cap Gris Nez.

It is noted that the Vancouver cable has a loss of 119 kW per 1 000 yd, which compares with 52 kW per 1 000 yd for the experimental cable used in the English Channel. The difference is, no doubt, largely due to the use of galvanized-steel armouring, but I agree with the authors that the superior mechanical characteristics of this material, as compared with aluminium alloy, justify its use.

Mr. R. J. Halsey: My colleagues and I in the Post Office are very pleased to have been associated with early sea trials that preceded this most important project. This was before the cable ship *Ocean Layer* was available, when it appeared that H.M.T.S. *Monarch* was the only suitable ship for the job. The *Monarch* was, however, fully engaged with the transatlantic telephone cable, which, with its polythene core only 0·6 in in diameter, also functions as a power cable of sorts. The test voltage was 90 kV d.c. or 310 kV/cm (maximum), and the breakdown strength was about 400 kV or 1 375 kV/cm (maximum). By comparison, the present cable was tested at a peak a.c. stress of 415 kV/cm and normally operates at about 120 kV/cm. I am not a power engineer and I wonder why a solid polythene dielectric cannot be used for submarine power cables. Is it a question of temperature rise? The construction is much simpler, and we already use polythene telephone cables approaching nearly 2 in diameter in shallow water.

I am glad to see that steel armour wires were used. Aluminium-alloy wires may exhibit a greater tendency to 'birdcage' in handling, and abrasion characteristics in tidal waters are much inferior. I am interested that the minimum permissible bending diameter is as low as 6 ft; much has been said about large-diameter laying gear, but when the Post Office considered the design of a new 4 000-ton cable ship it was concluded that the largest reasonable diameter for sheaves and cable drums was 7 ft.

Like M. Tellier, I wonder whether the ingenious method evolved for landing the shore ends really solves the problem for more open waters, where wind and tidal conditions may make positioning of the landing craft difficult. The narrow waters at Vancouver may be very different from the English Channel in this respect.

The speed of laying is given as approximately 3 knots and the surface currents at 4 knots. In the Post Office we feel that a laying speed of 3 knots is too low for the English Channel, and it will have to be raised to something like 5 knots, which may pose a problem.

Mr. C. H. Jolin: The tendency of the cable to twist could be avoided by the use of double wire armour with reverse lay, but the cable would be heavier and more difficult to coil.

With regard to the laying of the cable, the method using cable engines is very crude. I suggest that consideration be given to the use of a caterpillar control for laying the cable. I believe that one of the difficulties is the space it occupies, but in designing a new cable-laying ship one could have the ship's own machinery in the stern and use the whole of the rest of the space available for cable tanks. It might then be possible to find space for a suitable caterpillar.

When laying a cable on a sea crossing, speed is essential, and an appreciable proportion of the time is taken up in laying the shore ends. This could be reduced by having a cable joint just above the shore line, thus reducing the amount of cable to be handled on and off the scow, which takes an appreciable amount of the total laying time.

With reference to future work, it appears that d.c. transmission might be very largely used. I should like to know what type of insulation and what electrical stresses the authors would suggest for such cables. With d.c. cable it is obviously going to be a very important link. Is it really desirable to rely on only one cable where there is any chance of damage? For security of supply three cables might be used for a link, two as a go-and-return and one as a spare. Each cable would be smaller than a two-core cable designed for the same electrical stress and would be more easily handled.

Mr. H. R. Schofield: A previous speaker referred to the necessity for excellent seamanship and the finest supervision to carry out this immense project satisfactorily. I fully concur with that view, and will give one example. Let us imagine the *Ocean Layer*, aided by tugs, approaching the buoys for mooring off after laying the cable across the Channel. The cable appears to be a piece of string hanging over the bow sheaves. Any mismanipulation of the ship may easily cause a fault in the cable. Furthermore, the alignment of the scow, the fixing of the stopper and the bight extractor is a very difficult operation, and the cable gang on the scow must be very experienced. This operation was carried out admirably. Other difficult operations were at the landing end, where the scow was brought to one position where all the cables converged. I suggest that it would be better to space the landing cables apart a little. This would simplify the scow landing operations.

In laying the cable in one continuous length all the land cable is transferred to the scow. If joints were installed close in shore a number of operations would be eliminated and the installation costs would be reduced.

Mr. C. C. Barnes: Submarine power cables necessitate very careful consideration in regard to their mechanical design, since they are exposed to risks of sharp bends, heavy strain and vibration imposed during both the initial laying and any subsequent operations which may be necessary.

The C.E.A. first considered all the five types of proved super-voltage British power cable systems used for land transmission, but the vital need for long submarine power cables without joints in the initial laying operation resulted in a decision to carry out sea trials with a pre-impregnated gas-filled cable, as this design is the only one capable of being manufactured in one continuous length without joints.

Joint development work on the cross-Channel project began in November, 1952, and a sea trial was made off Dover from 21st July to 9th September, 1953. Further sea trials which

helped to prove the mechanical design, handling and laying of submarine power cables were as follows:

First Loch Fyne sea trial ..	16th–24th September, 1954.
Second Loch Fyne sea trial..	7th–16th December, 1954.
Falmouth sea trial	17th November to 10th December, 1955.

The primary objective of these sea trials was to determine the effect on the cable and associated joints of the severe mechanical conditions associated with the laying from a cable ship and to obtain experience in making and laying joints at sea.

Important lessons learnt were as follows:

(i) The length and direction of lay of the armouring, the diameter of wire and space factor have an important bearing on the way in which the cable coils down into the tanks (Fig. A) and on the possibility of kinking when handled on deck for jointing operations.

(ii) On the occasion of any damage to the rubber layer between the armouring wires and the reinforcing tapes, sea water may penetrate for an appreciable distance unless close and continuous contact can be maintained.

(iii) The mechanical strength and thickness of the reinforcing tapes need to be determined, with regard not only to the pressure within the cable, but also to the severe treatment (Fig. B) resulting when handling and lashing the cable (Figs. C and D). Rough handling of the cable near a flexible joint in the vicinity of a cut or free cable end will almost certainly cause longitudinal displacement between the conductor and the sheath, with resultant damage to the hand-applied dielectric, unless the conductor, sheath and armour are mechanically anchored together at the free end.

(iv) Joints at sea must be simplified so as to minimize the jointing time. A completion time of eight hours for joints made in the English Channel is very desirable.

(v) The storage capacity and equipment on the cable ship need very critical examination.

(vi) For a successful and expeditious installation, a high degree of co-ordination is essential between all personnel.

(vii) Long experience with telecommunication submarine cables is not, in itself, adequate for submarine power-cable installations where pressure cables are used which are highly stressed electrically.



Fig. A



Fig. B

With reference to Section 3 of the paper, what items are included under maintenance charges, and what annual fault rate is anticipated?

In Section 4.2 it is stated that, in 1955, over 32 000 vessels called at the Port of Vancouver alone and that trawling is not restricted in the Strait of Georgia. Some assessment of these hazards is therefore desirable.

In Section 7 it is stated that an important feature of submarine cable joints is the essential need to reduce the time for making them to the absolute minimum because of the weather hazard, and that a total time of 24 hours was achieved for rigid repair joints on the Vancouver cable. For a Channel installation about eight hours' total time is very desirable. Does the authors' experience indicate that it would be possible to meet this exacting requirement?

With reference to Section 8.2, what control tests were made on the pre-impregnated paper tapes and what limits were specified for these tests?

The accurate location of any faults is desirable. Will the authors explain the formula relating gas flow to the feed distance and the degree of accuracy which is obtained with this technique?

If a similar scheme occurred in the future would the authors favour the use of a high-voltage d.c. submarine cable?

Mr. W. Casson: Much experience was obtained by the cable company from sea trials carried out by the C.E.A. on a 132 kV a.c. cross-Channel cable interconnector. The design of the Vancouver cable was influenced by these trials, and when a sample length was tried out during further sea trials in Loch Fyne, it was found to be very superior to the sample tried out in the earlier sea trials.

Although a repair was made on one of the Vancouver cables in about 24 hours, I think that the location and repair of a cable fault in the middle of the English Channel might be a difficult and hazardous operation. I should like some further information on the accuracy of the gas-location methods described in Section 10.

The paper naturally raises the question of what is happening to the cross-Channel cable project. Sir Josiah Eccles has already given some information, and it may appear that progress has been slow. This has been partly due to the important decision which was recently made to change from a.c. to d.c. transmission and



Fig. C



Fig. D

which, I am sure, will have great influence on future large-scale submarine cable projects. The cable tenders for the cross-Channel scheme are due to be received by the end of May, and I wonder whether there will be an offer of a polythene insulated cable.

D.C. transmission, from a cable manufacturer's point of view, may not seem to be as remunerative as a.c. transmission, but I think that the development of the former will require cable in both sea and land schemes which, with an a.c. scheme, would be more costly and technically more difficult.

THE AUTHORS' REPLY TO THE ABOVE DISCUSSION

Dr. T. Ingledow and **Messrs. R. M. Fairfield, E. L. Davey, K. S. Brazier** and **J. N. Gibson** (*in reply*): We are grateful to all who took part in the interesting and valuable discussion and appreciate very much the remarks made in support of our work. Space does not permit detailed replies to all the questions raised, some of which we are obliged to deal with in a general manner.

In reply to Sir Josiah Eccles, the two supply undertakings serving Vancouver Island are interconnected and co-operate fully to their mutual advantage. The existence of two different systems on Vancouver Island thus did not affect the desirability or timing of the project. As a matter of fact, the cable not only supplies energy to the southern Vancouver Island system of the British Columbia Electric Company, but is being utilized to transmit substantial amounts of power on behalf of the utility serving the northern part of Vancouver Island. Thus, export to the latter reached a one-hour peak of 60 MW in April, 72 MW in May, and 94 MW in June, 1957.

The route selected for the submarine cable is considered to be the most suitable taking all relevant factors into account, including the nature of the sea bottom, present and anticipated load distribution, and available transmission routes across the mountainous country. At Sechelt, for instance, depths of 200 fathoms would be encountered and the nature of the sea bed is unfavourable. Also, it is not feasible to run a transmission line from station No. 12 in Fig. 1 in a more northerly direction because it would have to traverse over 100 miles of inaccessible territory, without roads, covered by mountains and glaciers anywhere between 6 000 and 9 000 ft high.

It is estimated that the cost of electricity delivered to Vancouver Island through the interconnection is about 80% of that obtainable from the installation of a modern coal or oil-fired steam generating station on the Island. This estimate includes operation and maintenance, depreciation and return on investment, and makes full allowance for the absorbed capacity of the existing system. The latter, of course, required an assessment of probable fault incidence and useful life of the submarine sections, which were also referred to by Mr. Barnes.

When gas pressure is applied to the cable the diametral strain imposed on the lead sheath is 1·2% (maximum) before the sheath is fully supported by the reinforcing tapes, having transferred its hoop stress to them. The sheath must be capable of withstanding the duty without failure, and the rate at which strain occurs under

operating conditions can be as low as 10^{-4}% per hour. The ultimate diametral strain (i.e. strain to fracture) of the unreinforced sheath, when subjected to internal gas pressure, decreases as the rate of strain falls owing to the complex intercrystalline nature of the fractures. For the sheath in question, the ultimate diametral strain for all rates of strain down to 10^{-4}% per hour is 4·0% (minimum), thus providing a factor of safety in excess of three in relation to service requirements.

We appreciate Sir John Dean's remarks with regard to the planning, engineering and cable design, and pay tribute to the excellent manner in which the captain and complement of the cable ship *Ocean Layer* carried out their part of the project. Our experiences confirm the merits of the Pleuger rudder for submarine cable laying.

We join with M. Tellier in expressing our profound sorrow at the passing of Maurice Laborde, who had taken such a deep interest in and had contributed much to the development of submarine power cables.

If the hollow-duct gas-filled cable is severed under water, gas will escape until the internal pressure is equal to the external water pressure. With these pressures balanced, water will penetrate along the duct only to a point where the cable rises by an amount equal to the duct diameter (approximately 0·5 in), to form a seal. Owing to the irregularity of the sea bottom even on general gradients, such points occur in practice at distances along the cable of a few yards; this distance is the order of the water penetration and has been confirmed by experiment.

The scow method developed for laying and landing the shore ends has, we consider, very definite advantages for certain conditions of installation (e.g. the Tsawwassen landing). We agree that the method would present difficulties under bad weather and sea conditions and that the methods must be adapted to the conditions associated with any particular installation. All our experience indicates, quite definitely, that the object should always be to lay the cable as a continuous length from shore to shore and that jointing at sea should be eliminated, except in cases where it cannot be avoided, e.g. to effect a repair. We agree with M. Tellier's proposals for making the closing joint in a repair.

In reply to Mr. Sayers, the timing of the project was governed essentially by the power needs of the Vancouver Island system. The engineering aspects were, however, greatly assisted by the experience gained from the work previously carried out for the C.E.A. in connection with the cross-Channel power-cable project, details of which have been recorded by Sayers, Laborde and Lane in Reference 1. High-voltage d.c. transmission was considered and rejected as uneconomic; also, insufficient practical experience was available at the time on the reliability of the converting equipment to justify its inclusion in a pioneering installation of this kind. It is not easy to compare the relative engineering difficulties associated with this project with those likely to be encountered on the cross-Channel scheme; the greater depth of the Georgia Strait section presented problems which will not be present in the Channel scheme, and the greater length involved in the latter presents no special difficulties. On the other hand, the sea and weather conditions in the Channel are likely to be worse, as also may be the hazard of damage from ships' anchors. Five cables were installed because of the pioneering nature of the project and the essential need of ensuring a firm supply to Vancouver Island; the weather conditions and availability of a cable-laying ship are such that long delays could occur before a cable failure could be repaired in the deep water of Georgia Strait. Only two additional cables will be required to provide the second circuit. Mr. Sayers refers to the need to develop a simpler design of submarine repair joint and Mr. Barnes suggests the time for jointing at sea should be reduced to eight hours. We agree that these objectives are highly desirable, and development work will continue with these ends in view.

Mr. Halsey asks why it is not possible to use a cable with a solid polythene dielectric. Under a.c. conditions the maximum stress at the conductor would have to be limited to about 30 kV/cm to avoid deterioration of the dielectric due to electrical discharge; such a cable (diameter over dielectric of about 10 in) would have been quite uneconomic and too large to handle. The possibilities of polythene-insulated submarine power cables for d.c. operation are, however, more promising and are, at present, in the development stage. We would limit the maximum conductor temperature of the polythene cable to approximately 60° C. Our observations on the use of the scow method of laying shore ends in the English Channel are dealt with in our reply to M. Tellier, and we foresee no serious difficulty in laying at a speed of 5 knots.

We agree with Mr. Jolin that the replacement of the conventional ship's cable engine by a caterpillar system is a desirable development for the future, and with him and Mr. Schofield that by jointing the cable on land, above the water line, the difficulties of landing the shore ends can be materially simplified. For d.c. transmission, which, as Mr. Jolin states, is likely to develop in future to handle increasing power over longer distances, the present design of cable is the only type we consider to be satisfactory and adequately proved in all respects. As has been stated in reply to Mr. Halsey, polythene-insulated cable is under development, and it may be that this type will be developed successfully to the practical stage, in which case the assessment of the two types will depend on their relative technical and economic merits. The d.c. electrical stresses for both types will be of the order of at least 140 kV/cm (mean), or 240 kV/cm (maximum), based on the logarithmic formula. We agree that a d.c. system employing more than one cable is advantageous in ensuring a firm supply.

In reply to Mr. Schofield, the cables were landed close together to reduce installation costs, but we agree that spacing of the landings, where conditions allow it, would simplify operations.

Mr. Barnes reminds us of some of the valuable information learned from the cross-Channel sea trials carried out on behalf of the C.E.A. and dealt with in Reference 1, and of the Loch Fyne sea trials, which were part of the proving programme of the present project. The control tests of the pre-impregnated paper tapes were very comprehensive, but space does not permit full details being given here. They included mechanical, dielectric, physical and chemical properties; very close limits were specified and measurements were taken and recorded throughout manufacture. The gas-leak location technique, mentioned also by Mr. Casson, is again too lengthy to be described in our restricted reply, but the information is available and we consider the accuracy of location to be within $\pm 2\frac{1}{2}$%. The economics of the interconnection would be in favour of alternating rather than direct current for as far ahead as we can reasonably foresee.

Most of the points raised by Mr. Casson have been dealt with in the replies to other speakers. In connection with the C.E.A. requirements for the cross-Channel scheme, designs of polythene-insulated cables will be offered. Unlike the gas-filled design, however, further development work and laying trials at sea have to be completed before we can recommend the polythene design as a practical engineering proposition.

621.319.4 : 621.315.1

The Institution of Electrical Engineers
Paper No. 2252 S
Jan. 1957
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VOLTAGE PERFORMANCE OF SERIES CAPACITORS IN TRANSMISSION AND DISTRIBUTION LINES

By J. M. MAGOWAN, Associate Member.

(*The paper was first received 9th April, and in revised form 25th September, 1956. It was published in January, 1957, and was read before the* SUPPLY SECTION *22nd May, 1957.*)

SUMMARY

The paper is divided into two Sections, the first of which deals with the advantages of series capacitors for voltage control purposes, and suggests the possibility of dispensing with the use of capacitor protective equipment in the case of certain types of distribution line. The second Section describes existing installations in 11 kV and 88 kV lines.

(1) INTRODUCTION

The Electricity Supply Commission of Southern Rhodesia is a corporate body responsible for supplying electricity to all municipalities in the Colony (with the exception of Salisbury and Bulawayo), to extensive gold and base-metal mining areas and to vast rural areas comprising townships, villages and farms, altogether extending over an area approximating to that of England and Scotland (Fig. 1).

Development of the system since the Commission's inception in 1936 can be regarded as phenomenal, resulting now in voltage-drop conditions on the transmission and distribution network which are presenting a considerable problem in providing supply to widely dispersed consumers.

Normal methods of improving voltage regulation, such as automatic on-load tap-changing transformers and induction regulators, are being used; recently, however, a decision was taken to improve voltage regulation by means of capacitors in series with selected lines.

There are two more or less distinct main types of application for series capacitors in electric power lines: one is their use in long-distance tie-lines for the purpose of reducing transfer reactance and thus raising the stability level of the system; the other is concerned with distribution lines where the object is to reduce voltage drop by reduction of line reactance. The paper is concerned only with voltage performance of series capacitors in lines, and comparisons are drawn between the performance of shunt and series capacitors for boosting voltage.

(2) EFFECT OF CAPACITORS ON LINE RECEIVING VOLTAGE

(2.1) Voltage Performance of Lines

For lines in which susceptance may be neglected, the voltage drop on one phase is given with reasonable accuracy by the approximate expression

$$V = IR\cos\phi_2 + IX_L\sin\phi_2 \quad . \quad . \quad . \quad . \quad (1)$$

where I = Line current.
R = Line resistance, ohms.
X_L = Line reactance, ohms.
$\cos\phi_2$ = Power factor of the load.

The receiving voltage at the end of the line (Fig. 2) is

$$V_2 = V_1 - V = V_1 - I(R\cos\phi_2 + X_L\sin\phi_2) \quad . \quad (2)$$

When no form of capacitor voltage correction exists

$$-\frac{\partial V_2}{\partial I} = Z = R\cos\phi_2 + X_L\sin\phi_2 \quad . \quad . \quad . \quad (3)$$

Capacitors shunting load centres reduce the voltage/current phase displacement and thus the voltage drop. Capacitors in series with lines supplying inductive loads reduce voltage drop by compensating the inductive reactance of the line.

(2.2) Shunt-Capacitor Voltage Performance

For a given load, the resistive component of voltage drop remains constant as the load power factor is raised, but the inductive component becomes less. If the voltage/current phase displacement is reduced from ϕ_2 to ϕ_3, the receiving voltage at the end of the line becomes

$$V_2 = V_1 - V = V_1 - IX_L\left[\left(\frac{R}{X_L}\cos\phi_2 + \sin\phi_2\right) - (\sin\phi_2 - \cos\phi_2\tan\phi_3)\right] \quad . \quad (4)$$

Therefore

$$-\frac{\partial V_2}{\partial I} = Z' = X_L\left[\left(\frac{R_L}{X_L}\cos\phi_2 + \sin\phi_2\right) - (\sin\phi_2 - \cos\phi_2\tan\phi_3)\right] \quad . \quad (5)$$

and, subtracting eqn. (5) from eqn. (3),

$$Z - Z' = X_L(\sin\phi_2 - \cos\phi_2\tan\phi_3) \quad . \quad . \quad . \quad (6)$$

When $\cos\phi_3 = 1$, $\cos\phi_2\tan\phi_3 = 0$, and the reactive component of voltage drop is then eliminated. If $\cos\phi_3 < 1$ leading, $\cos\phi_2\tan\phi_3$ is negative, and V_2 therefore rises as additional capacitors shunt the load. When

$$R\cos\phi_2 + X_L\sin\phi_2 = X_L(\sin\phi_2 - \cos\phi_2\tan\phi_3)$$

the total voltage drop is eliminated and $V_1 = V_2$.

The change in voltage effected by shunt capacitors always increases the value of V_2, and the voltage boost due to them is

$$\text{Phase boost} = IX_L(\sin\phi_2 - \cos\phi_2\tan\phi_3) \quad . \quad . \quad (7)$$

But

$$I = \frac{P}{V_2\cos\phi_2}$$

where P is the power per phase, and therefore

$$\text{Phase boost} = P(\tan\phi_2 - \tan\phi_3)\frac{X_L}{V_2} \quad . \quad . \quad (8)$$

The rating per phase of capacitors necessary to reduce voltage/current phase displacement from ϕ_2 to ϕ_3 is

$$Q = P(\tan\phi_2 - \tan\phi_3) \quad . \quad . \quad . \quad . \quad (9)$$

Mr. Magowan is with the Electricity Supply Commission, Southern Rhodesia.



Fig. 1.—Map of Southern Rhodesia showing the Electricity Supply Commission's licensed area of supply.

Thus $$\frac{\text{Phase boost}}{Q} = \frac{X_L}{V_2} \quad . \quad . \quad . \quad . \quad . \quad (10)$$

Shunt-capacitor voltage boost is thus independent of load magnitude and power factor. For a given capacitor it is proportional to the inductive reactance of the line and inversely proportional to the line receiving voltage.

With capacitors solidly connected at load centres, no improvement in voltage regulation results; in fact, the opposite effect prevails because the rating of a capacitor is proportional to the square of the applied voltage. The presence of shunt capacitors thus serves to accentuate changes in voltage at the load centre.

In order to improve regulation with shunt capacitors it is necessary to employ ancillary equipment to switch them in sympathy with load reactive current or with receiving voltage. Alternatively, capacitors can be associated with consumers' equipment and connected on the load side of switches controlling such equipment. In such circumstances, however, the capacitors are not under the direct control of the supply authority.

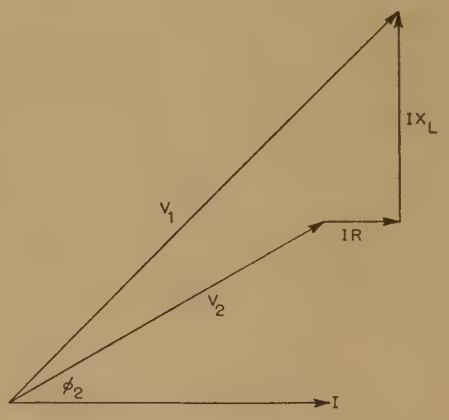


Fig. 2.—Receiving voltage for line without series capacitors.

(2.3) Series-Capacitor Voltage Performance

Owing to its inherently low loss, the capacitor can be regarded as an almost purely reactive device, and its presence in a line does not appreciably alter the active current component in the line. On the other hand, lagging current develops a quadrature voltage which is in phase opposition to the inductive voltage drop in the line.

The voltage drop on one phase of a line is

$$V = IR\cos\phi_2 + I(X_L - X_C)\sin\phi_2 \quad . \quad . \quad . \quad (11)$$

where X_C is the reactance of the series capacitor. The receiving voltage (Fig. 3) is

$$V_2 = V_1 - V = V_1 - I[R\cos\phi_2 + (X_L - X_C)\sin\phi_2] \quad (12)$$

$$= V_1 - I\left(\frac{R}{\tan\phi_2} + X_L - X_C\right)\sin\phi_2 \quad . \quad . \quad . \quad . \quad (12a)$$

The change in voltage due to the series capacitor is

$$-\frac{\partial V_2''}{\partial I} = Z'' = R\cos\phi_2 + (X_L - X_C)\sin\phi_2 \quad . \quad (13)$$

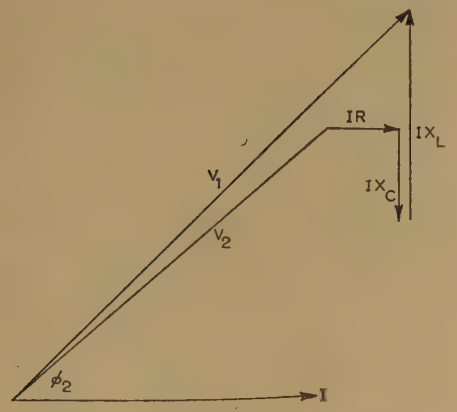


Fig. 3.—Receiving voltage for line with series capacitors.

Subtracting eqn. (13) from eqn. (3),

$$Z - Z'' = X_C \sin \phi_2 \quad . \quad . \quad . \quad . \quad . \quad (14)$$

If $\cos \phi_2 = 1$, then $\sin \phi_2 = 0$, and the series capacitor has no effect upon line voltage. If $\cos \phi_2 < 1$ leading, then $Z - Z'' = X_C \sin \phi_2$ becomes negative and causes a decrease in voltage rise. In this way series capacitors will reduce the voltage rise experienced at the end of long unloaded lines.

The practical case with which we shall be concerned is when $\cos \phi_2 < 1$ lagging, and $Z - Z'' = X_C \sin \phi_2$ is positive, causing a rise in voltage. The line reactive drop is eliminated when $X_C = X_L$; when $R/\tan \phi_2 + X_L = X_C$ the total voltage drop is eliminated. In the latter instance the inductive reactance of the line is over-compensated.

In the practical case with a series capacitor of reactance X_C,

$$\text{Phase boost} = IX_C \sin \phi_2 \quad . \quad . \quad . \quad . \quad (15)$$

and, since the rating per phase of the series capacitor is

$$Q = I^2 X_C \quad . \quad . \quad . \quad . \quad . \quad . \quad (16)$$

then

$$\frac{\text{Phase boost}}{Q} = \frac{\sin \phi_2}{I} \quad . \quad . \quad . \quad . \quad . \quad (17)$$

Boost is thus independent of line reactance and voltage but is dependent upon load magnitude and power factor.

(2.4) Comparison between Shunt and Series Capacitors

It is seen from eqns. (10) and (17) that the voltage boost due to compensation by shunt or series capacitors is in each case directly dependent upon capacitor rating. Calling the degree of compensation α, we will ascribe to it unity value when full compensation of line reactive voltage drop is achieved. With series capacitors in a line supplying a given load, the capacitor rating

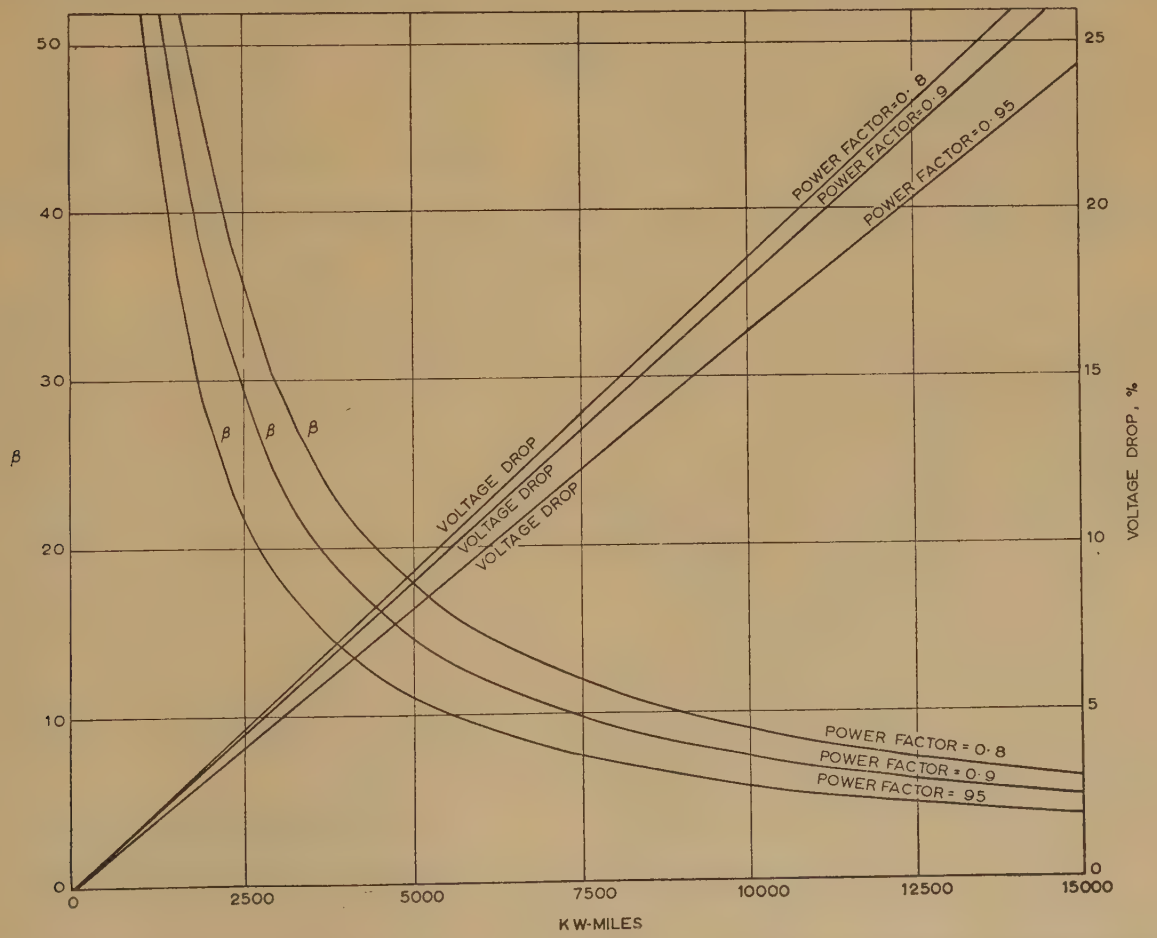


Fig. 4.—Ratio of shunt to series capacitor rating for equivalent voltage boost with 11 kV line having 0·025 in² conductor of 48 in spacing.

Resistance per mile = 1·75 ohms.
Reactance per mile = 0·65 ohm.

increases as its reactance X_C is raised, unity compensation being obtained when $X_C = X_L$. In the case of capacitors shunting a load, their rating increases as $\tan\phi_2 - \tan\phi_3$ becomes larger, unity compensation being achieved when $\tan\phi_3$ is zero. For any degree of compensation by series or shunt capacitors we have

$$\alpha = \frac{X_C}{X_L} \quad \text{for series compensation,}$$

and

$$\alpha = 1 - \frac{\tan\phi_3}{\tan\phi_2} \quad \text{for shunt compensation.}$$

Calling the ratio of the rating of a shunt to a series capacitor β for a given degree of compensation, we have, by combining and rearranging eqns. (10) and (17),

$$\frac{Q_{SH}}{Q_{SER}} = \beta = \frac{V_2^2 \cos\phi_2 \sin\phi_2}{PX_L} \quad . \quad . \quad . \quad (18)$$

$$= \frac{Z_L \sin\phi_2}{X_L} \quad . \quad . \quad . \quad . \quad . \quad (19)$$

where Z_L is the phase impedance of the load.

Fig. 4 shows β plotted against kilowatt-miles for an 11 kV line of 4 ft equilateral spacing using 0·025 in^2 conductor. Also plotted against kilowatt-miles is the percentage total voltage drop for loads of various power factors. Fig. 5 gives similar information for an 11 kV line of 4 ft equilateral spacing but with 0·05 in^2 conductor. Fig. 6 is concerned with a 33 kV line of 6½ ft equilateral spacing with 0·075 in^2 conductor, and Fig. 7, with an 88 kV line of 11 ft equivalent delta spacing and 0·20 in^2 conductor. Examination of eqns. (18) and (19) and these curves reveals:

(*a*) Being dependent upon $\cos\phi_2 \sin\phi_2$, β is a maximum when the load power factor is 0·7, and over the power-factor range 0·6–0·8 it will be within 4% of this maximum.

(*b*) For loads of the same power factor, whether heavy loads supplied over short lines or light loads supplied over long lines, β will have the same value for equivalent kilowatt-mile loading.

(*c*) At a given spacing, a heavier-section line has lower inductance and thus a higher value of β. For given kilowatt-miles, however, such lines experience less total voltage drop.

The value of β indicates the technical superiority of series over shunt capacitors for boosting voltage, and becomes extremely high for low kilowatt-mile loading with loads having a power factor in the region of 0·7. There is, nevertheless, still a considerable advantage to be gained in the case of heavy loads of higher power factor supplied over long-distance h.v. lines.

(2.5) Over-Compensation

With lines of small cross-section the resistive component is usually much greater than the reactive component of voltage

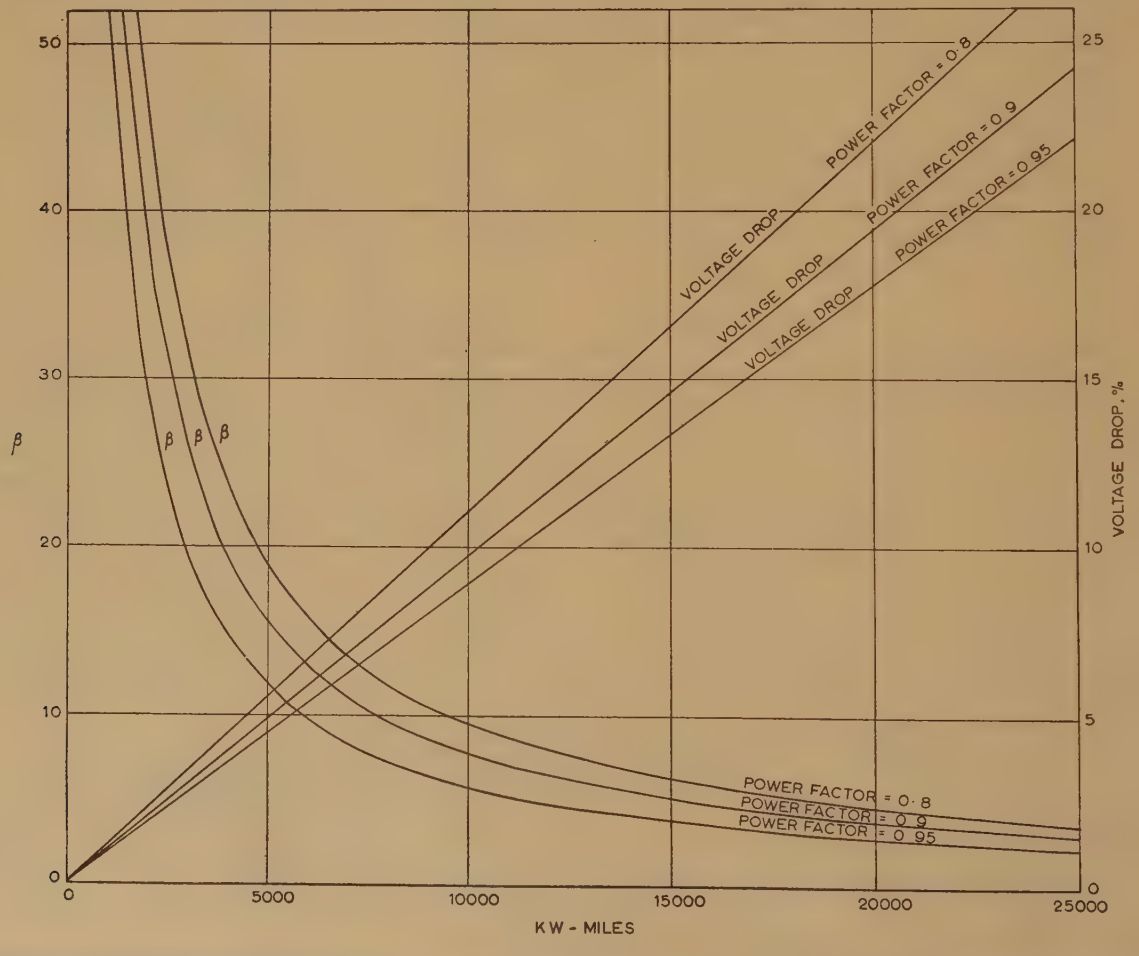


Fig. 5.—Ratio of shunt to series capacitor rating for equivalent voltage boost with 11 kV line having 0·05 in^2 conductor of 48 in spacing.

Resistance per mile = 0·87 ohm.
Reactance per mile = 0·615 ohm.

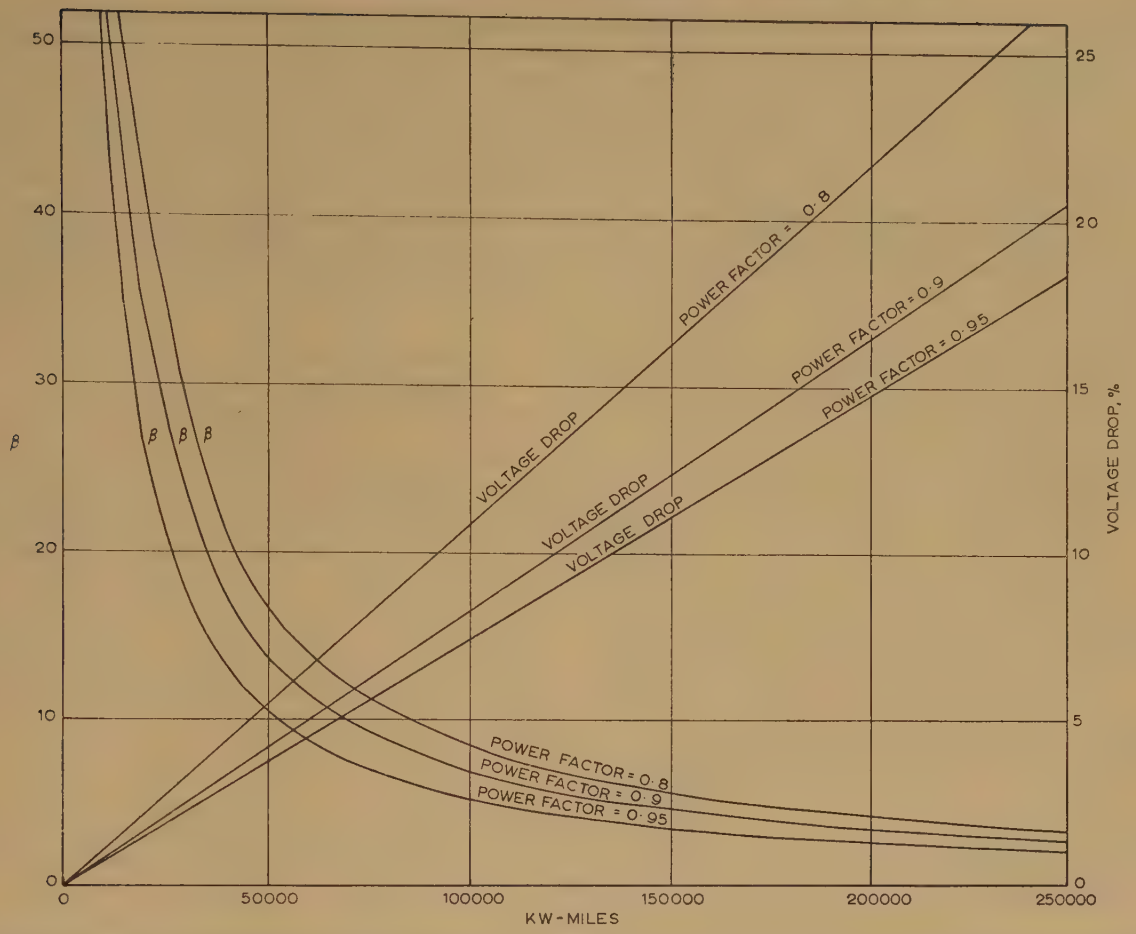


Fig. 6.—Ratio of shunt to series capacitor rating for equivalent voltage boost with 33 kV line having 0·075 in² conductor of 78 in spacing.

Resistance per mile = 0·60 ohm.
Reactance per mile = 0·617 ohm.

drop. Such lines are often not heavily loaded, and this, combined with a relatively low reactance, makes the value of β considerable. Over-compensation of such lines by series capacitors would not therefore be costly in capacitor rating as would be the case with shunt capacitors.

If over-correction is carried out with shunt capacitors, it results in leading-power-factor loads, and the chief advantage of shunt correction, the reduction of line current, is lost. Furthermore, lines operating at a leading power factor could be an embarrassment to the system as a whole. With over-correction of such lines by series capacitors neither of these disadvantages would result, and an automatic boost compensating for all of the reactive drop and part or all of the resistive voltage drop could be economically achieved. Over-compensating by series capacitors therefore offers an interesting field for experiment and is likely to prove a very useful method of dealing with small-conductor lines with their high resistive voltage drops.

(3) SERIES-CAPACITOR OPERATION

(3.1) Series-Capacitor Protection

A series capacitor carries line current which develops a voltage IX_C across it. Under fault conditions, a series capacitor would be called upon to carry currents larger than normal, and the voltage stress on the capacitor would be proportionately higher. Arrangements must therefore be made to preclude the possibility of capacitors suffering damage from over-voltage due to fault currents.

It is usual, where series capacitors are used for voltage control, to employ an arc-gap, which in turn actuates a contactor shunting the capacitor. Arc-gaps may be set to break down when voltage on the capacitors is from 1·5 to 2·5 times normal full-load voltage rating. The gap setting chosen would depend upon the type of load supplied; for example, assuming that one large motor formed a major part of the load on a line, its starting current would necessitate a high gap setting if the capacitor protection equipment were not to operate each time the motor started.

A more satisfactory and perhaps cheaper form of capacitor protection has been developed in America.[1] It employs an arc-gap in which the arc is extinguished each half-cycle. The gap is designed to operate for the duration of the fault without impairing gap characteristics. Thus the series capacitors are automatically reinserted in the line within a half-cycle of fault clearance. This feature is useful where series capacitors are used to raise the stability limit of a system, because transfer reactance is rapidly reduced after fault clearance. Where series capacitors are used to improve voltage regulation this feature is by no means essential.

With series capacitors used for automatic voltage control it is not always necessary to use spark-gap and contactor protection, since the voltage time to breakdown characteristic of the capacitor dielectric can, in some cases, be relied upon to afford protection until the normal line protective equipment operates.

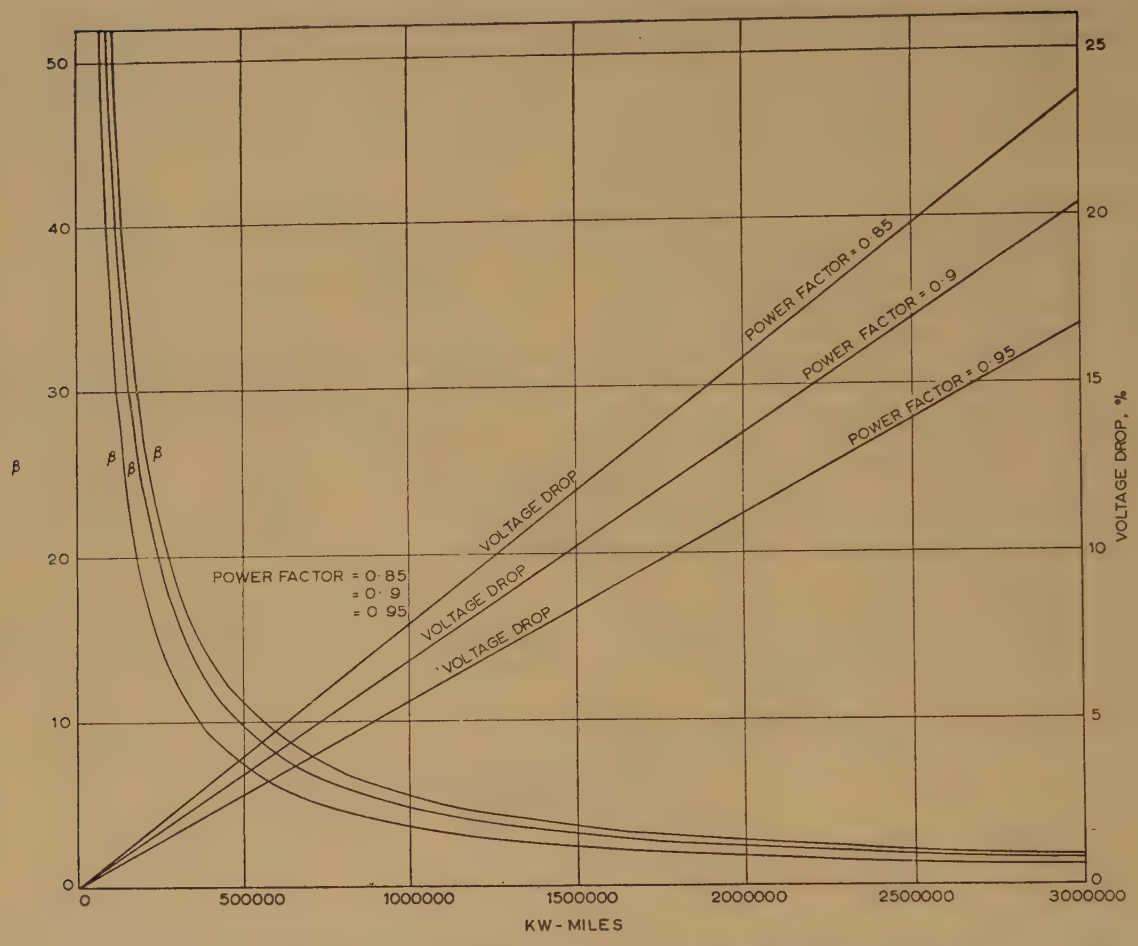


Fig. 7.—Ratio of shunt to series capacitor rating for equivalent voltage boost with 88 kV line having 0·2 in² conductor of 132 in spacing.

Resistance per mile = 0·226 ohm.
Reactance per mile = 0·624 ohm.

The chief factors influencing a decision to dispense with spark-gap and contactor protection are:

(*a*) Value of β for line in question.
(*b*) Fault current with capacitor limiting circuit reactance.
(*c*) Operating speed of line protective equipment.

Where β is high the line current is usually small; this limits the nominal rating of a series capacitor to a low value, and it becomes economical to under-rate the capacitor. For instance, three 10-ohm capacitors in an 11 kV 3-phase line supplying a 500 kW load at 0·8 power factor would each have a nominal full-load rating of 10·9 kVAr. Rating the capacitors at 24·5 kVAr would permit 150% normal full-load line current before the capacitors were stressed to their nominal voltage.

The cost of under-rating such small capacitors is much less than that of providing arc-gap and contactor protective gear. Also, the full-load voltage across the capacitor in applications of this type would be so low as to complicate the design of a gap for direct connection to the capacitor.

Lines having low values of β, e.g. an 88 kV line supplying a heavy load over a long distance, would require nominally rated capacitors in the three lines totalling several thousand kilovars. It would obviously be cheaper to employ other means of protection than under-rate such capacitor installations.

With lines of high β, the degree of capacitor under-rating to be employed is governed by the maximum fault current existing with the capacitor in circuit. The operating characteristics of line protective equipment would of course modify the degree of capacitor under-rating required.

(3.2) Resonance Effects with Series Capacitors[2]

The tendency for sub-synchronous resonance to occur appears to have been somewhat overstressed in recent literature,[3] and a suggested method of preventing it involves the use of resistors shunting the capacitors. The resistance recommended is ten times the capacitor reactance.

So far as damping effect on oscillatory currents is concerned, this is similar to including resistors of one-tenth the value of the capacitor reactance in series with the capacitors. This represents a very small increase in total circuit resistance, particularly with lines of small cross-section, and would not have any large effect upon damping.

For a motor to exhibit a tendency to sub-synchronous resonance its power rating would require to be a considerable part of the line rating. Practical factors militate against this possibility because only with small-section lines is a given motor likely to form an appreciable part of the total load, and in such cases circuit resistance is high.

(3.3) Ferro-Resonance

When supplied by a line containing series capacitance, initial heavy currents can continue as a resonant condition. The

resonant frequency is low and can develop high voltages across the capacitor. Ferro-resonance, being associated with unloaded transformers when first energized, can be avoided if the transformers are loaded. When the circuits supplying the transformer contain sufficient resistance to heavily damp oscillatory currents ferro-resonance cannot occur. The factors mentioned in connection with the effect of circuit resistance in relation to sub-synchronous resonance of motors apply equally to sub-synchronous resonance of transformers.

It is preferable that resistors are not used to shunt series capacitors because their presence causes power loss. With large capacitor installations on lines of low β the continuous loss could be considerable.

(4) LEMBWE INSTALLATION: 11 kV LINE

(4.1) System Conditions

Owing to the increasing demands for power from mining consumers along the Chrome Dyke in the Umvukwes area of Southern Rhodesia, the voltage conditions at the end of the 11 kV radial feeder serving this area had been deteriorating for some time. Towards the end of 1952, and before it was possible to extend the 33 kV system into the area, it became apparent that immediate steps would have to be taken to improve the voltage regulation, which at that time was of the order of 25%.

Considerable attention having been focused on the advantages to be obtained from the installation of capacitors in series with such lines, it was decided to make an experimental installation.

(4.2) Supply Arrangements

The supply to this area is derived from a 33/11 kV main step-down substation (m.s.d.s.) at Lembwe, which is served by means of a 33 kV line from Salisbury, a distance of 56 miles (Fig. 8). At Lembwe m.s.d.s. there are installed three 500 kVA 33/11 kV automatic on-load tap-changing transformers capable of providing a voltage variation at the secondary terminals of +15% to −6% in 14 steps of 1·5% each. All three transformers are capable of being operated in parallel.

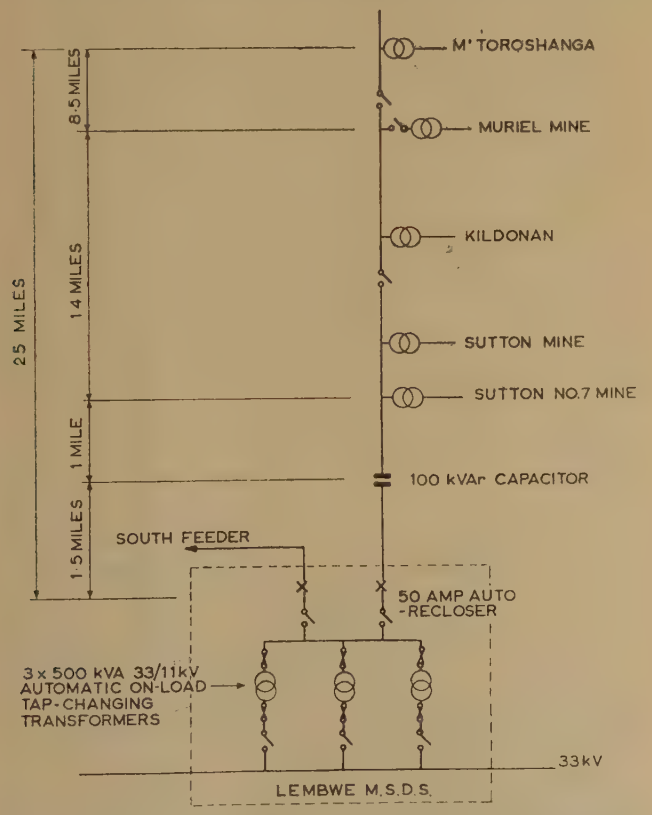


Fig. 8.—Diagrammatic illustration of the Lembwe–M'toroshanga 11 kV 0·05 in² line.

The capacitors were installed in the North feeder at a point approximately 1·5 miles from Lembwe m.s.d.s. The total length of this feeder at the date of the installation of the capacitors was 40 miles, excluding tee-offs.

For the purpose of the paper it is proposed to consider only the effect of the capacitor installation on the voltage regulation at M'Toroshanga. The section of the line between Lembwe m.s.d.s. and this point consists of 20 miles of 0·05 in² copper conductor erected in wishbone formation of 4 ft spacing. The β-value of the line and total load viewed from Lembwe was 6·5 at the time the capacitors were installed.

The feeder is protected at its source by means of an auto-recloser rated at 50 amp c.m.r. and a minimum tripping current of 100 amp. The opening sequence of the auto-recloser is two fast and two retarded operations prior to lock-out.

(4.3) Capacitor Rating

It was decided to employ existing 3-phase capacitors of 550 volts 100 kVA rating. When reconnected internally these produced capacitors of 12 ohms reactance with a nominal current rating of 96 amp.

The normal rating applied to power capacitors, namely voltage and kilovar rating, is not well suited to series capacitor applications: it is suggested that reactance and current rating would be more convenient.

Actual full-load current in the North feeder at the time of the installation was 50 amp at a power factor of 0·8, and 600 volts would be developed across the capacitor. This means that an overload of approximately 180% would stress capacitors at only their normal voltage rating.

Operational experience has shown that, with the type of line protection equipment employed, the capacitors may not have required to be under-rated to anything like the extent which they were. For example, there were periods during the summer rains when the peak loads built up to 82 amp at Lembwe, and this resulted in the capacitors being stressed to within 10% of nominal voltage rating.

This installation is of an experimental nature and is, in fact, the first to be used in conjunction with conventional lines on the African continent. It was therefore decided to adopt caution and to under-rate the capacitors, since no information was available concerning the performance of series capacitors under conditions where no protective equipment was provided.

(4.4) Operation

The capacitors were installed in April, 1953, and energized immediately afterwards.

The area loading conditions were such that it had previously been necessary to adjust the control circuits of the automatic on-load tap-changing equipment at Lembwe m.s.d.s. so that it operated on the highest tap position for the greater part of the 24-hour period. After the installation of the capacitors, however, it was possible to utilize a far greater range of the tap-changing gear by reducing the control rheostat setting. As an illustration of the greater range available since the installation of the capacitors, the following figures of the average number of tap changes per day are quoted:

Before capacitor installation, 4·6.
After capacitor installation, 26.

Recording instruments for current were installed at Lembwe

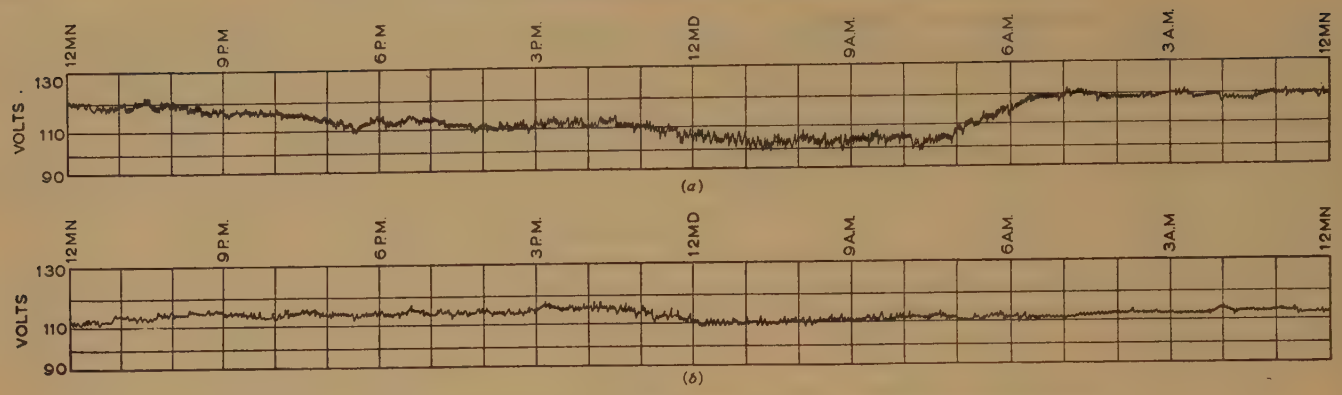


Fig. 9.—Voltage curves before and after installation of Lembwe series capacitor.
(*a*) Before. (*b*) After.

m.s.d.s., and for voltage at the M'Toroshanga substation, prior to and after the series capacitor installation. Examination of the voltage charts taken at M'Toroshanga revealed that an improvement in the effective voltage regulation has been attained from approximately 24·5% to approximately 9·25% (Fig. 9). The improvement in line-to-line regulation due to the capacitors alone would be approximately 7·0%, but, in addition, their installation has brought the line within the scope of the automatic on-load tap-changing transformer.

It will be of interest to note that no shunt resistors are fitted to the capacitors for reducing the possibility of sub-synchronous or ferro-resonance. During commissioning it was arranged that all consumers' load was off and the transformers were energized under this condition. It was further arranged that the largest motor on the line (150 h.p., 960 r.p.m.) at Muriel Mine was started under load on its own. No resonant conditions occurred.

The installation has now been in commission for four years, and it may be concluded from the experiences gained that the feasibility of dispensing with arc-gap and contactor capacitor protection and shunting resistors has been established for this type of line. It is also noteworthy that the capacitors have suffered no damage in an area which experiences severe lightning conditions during summer months.

(5) PENHALONGA INSTALLATION: 11 kV

(5.1) System Conditions and Supply Arrangements

At Stapleford Forest in the Eastern Districts of Southern Rhodesia a sawmill is supplied by 15 miles of 0·025 in^2 copper line from a m.s.d.s. containing two 500 kVA automatic on-load tap-changing transformers. The m.s.d.s. is itself supplied by 16 miles of 33 kV line from the power station at Umtali. Other loads supplied by this m.s.d.s. are given in Table 1, and the distances from the m.s.d.s. to the tee-offs are indicated in Fig. 10, which also shows where the series capacitor is installed.

Table 1

LOADS SUPPLIED FROM PENHALONGA M.S.D.S.

	kW
Penhalonga–Mountain Home	513
Mountain Home–Odzani	477
Odzani–Sheba	455
Sheba–Stapleford	360

The 0·025 in^2 line is protected at its source by an auto-recloser of a type similar to that at Lembwe. The continuous current rating is 40 amp and minimum tripping current 80 amp. The opening sequence of the recloser is two fast and two slow operations prior to lock-out.

With loads such as sawmills the instantaneous demand can

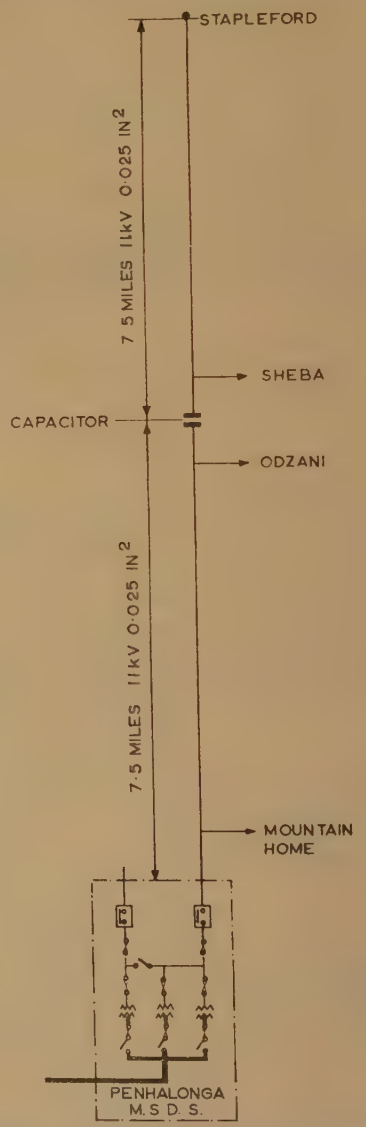


Fig. 10.—Diagrammatic illustration of the Penhalonga–Stapleford 11 kV 0·025 in^2 line.

greatly exceed the half-hour demand indicated on the normal type of maximum demand meter. When supplied by several miles of 0·025 in^2 line, this leads to large voltage drops of short duration which in turn are liable to cause lamp flicker.

Sawmills are notorious for operating at low power factor and this does much to aggravate voltage conditions. But at this mill a degree of correction has been applied and at full load the power factor is 0·84. It should be noted, however, that the full loads indicated in Table 1 are based upon half-hour demand.

(5.2) Capacitor Rating

Since the resistance/reactance ratio of the line is as high as 2·8 and the sawmill load operates at 0·84 power factor, the voltage drop on the line is predominantly resistive. With full compensation of the line inductive reactance by series capacitors, the improvement in voltage regulation would not be worth while. It was therefore decided to employ over-compensation.

In view of the experience gained with the Lembwe installation, and having regard to the types of 11 kV lines operating and projected in Southern Rhodesia, it was desired to select a capacitor rating which could be widely applied. The choice was for capacitors of 10-ohm 40 amp rating, which could be used singly or in series on lightly loaded lines and in series-parallel combinations on lines having heavier loading.

It was decided to install three such capacitors per phase in series in this line (as shown in Fig. 11). By opening short-circuiting links across the capacitor terminals, a total capacitive reactance of 30 ohms could be introduced. The inductive reactance of the line is 9·4 ohms so that over-compensation approaching 320% would be achieved.

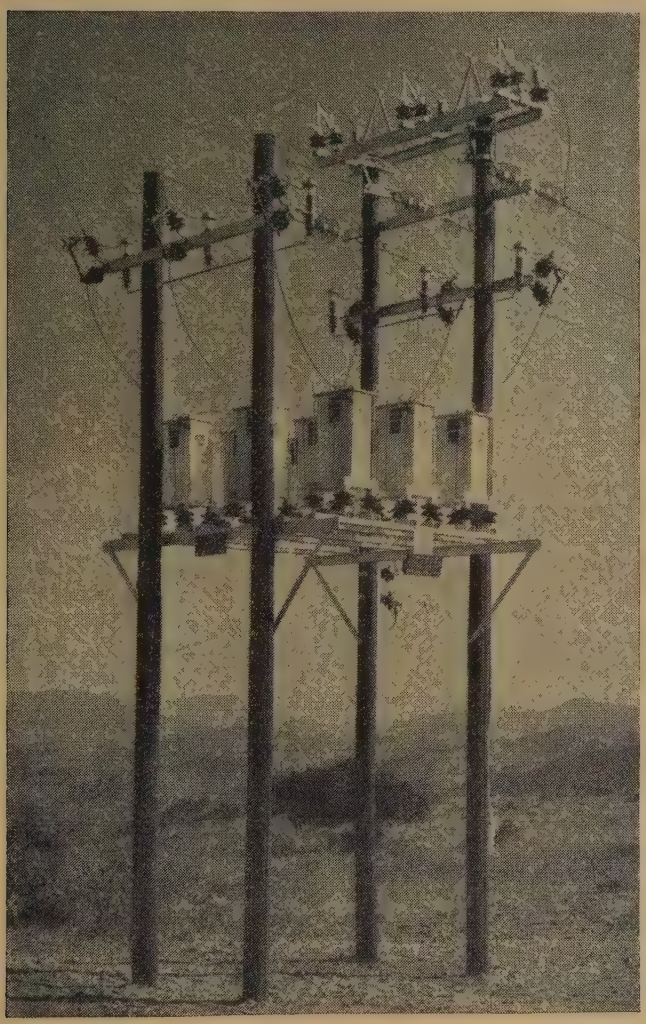


Fig. 11.—View of the series capacitor installation at Penhalonga.

(5.3) Operation

The installation was commissioned during July, 1956, first with 10 ohms, then with 20 ohms, and finally with 30 ohms of capacitive reactance per phase. It is intended that the three capacitors per phase remain in circuit permanently.

For a month prior to the commissioning, voltage- and current-recording equipment was installed at Penhalonga m.s.d.s., and voltage-recording equipment at Stapleford. It was found that the day-to-day pattern of load and voltage performance was consistent.

Based upon the loads given in Table 1, the calculated line-to-line voltage performance of the installation is shown in Fig. 12 with 30-ohm series capacitors at various assumed load power factors. The voltage at Penhalonga is regarded as fixed at 105% nominal.

Before commissioning the capacitors, examination of the Stapleford charts revealed that the voltage variation under load conditions was from −1·6% to −7·5%, with the Penhalonga voltage maintained at 105% nominal. The regulation of 12·5% applies to the Stapleford end of the line. That at the sawmill busbar would be 16%, owing to the reactance of the consumer's substation transformer.

Based upon the voltage variation under load, with no series capacitors installed, the calculated effect of capacitor installations ranging up to 50 ohms per phase is given in Fig. 13. Voltage records taken when capacitors of 10, 20 and 30 ohms per phase were installed are marked on Fig. 13 and indicate the discrepancy between calculated and observed performance.

Upon energizing the capacitor in 10-, 20- and 30-ohm stages it was arranged that no load existed on the line, in order to determine whether ferro-resonance would occur, but no resonances occurred under these conditions. Furthermore, for each stage of capacitor compensation it was arranged that the first load imposed on the line would be the largest individual motor in the sawmill (175 h.p., 960 r.p.m.). The motor started under load in normal fashion with each stage of capacitor correction on the line and exhibited no tendency to lock-in at a reduced speed.

(6) HARTLEY INSTALLATION: 88 kV

(6.1) System Conditions and Supply Arrangements

From the Commission's main generating station at Umniati, lines extend south to Gwelo and north, via Gatooma and Hartley, to Norton, where feeders radiate east and north-east to Salisbury City and Mazoe.

The length of line between Umniati and Norton is 90 miles, and the radial feeders to Mazoe and Salisbury City are 33 miles and 23 miles, respectively (see Fig. 14). The conductor size used throughout is 0·2 in^2 steel-cored aluminium.

Forecasts indicate that by 1959–60 the load centred upon Norton will reach a total of 25 MW at a power factor of approximately 0·85. This condition would result in a voltage drop at this point approaching 25%, and, as the maximum over-voltage available on the 88 kV busbars at Umniati is 5%, Norton 88 kV busbars will experience a full-load voltage of 20% below nominal rating, resulting in excessive voltage drop at the terminal busbars of the radial feeders.

Although it is not expected that serious voltage difficulties are likely to arise for two or three years, arrangements were made for the installation of a series capacitor in this 88 kV line, and it was commissioned during August, 1956.

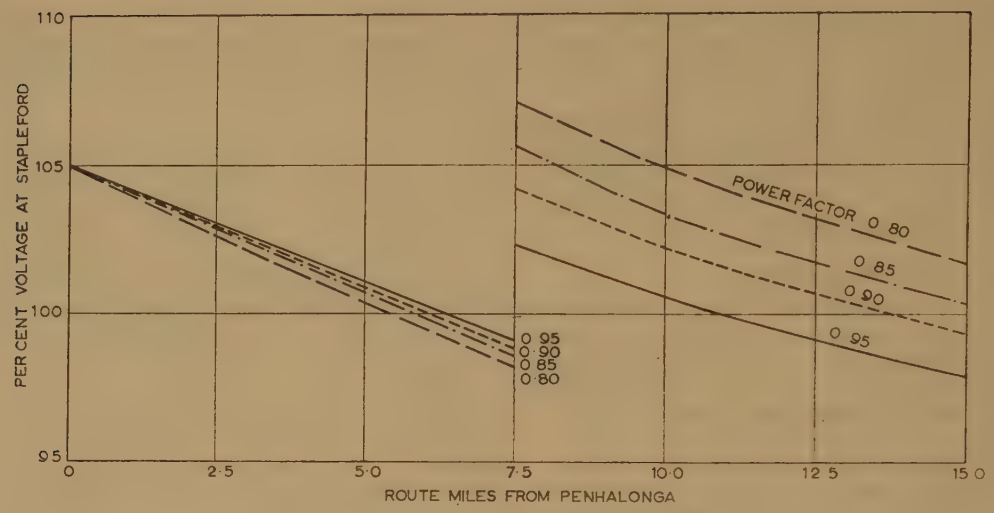


Fig. 12.—Calculated performance of Penhalonga series capacitor.

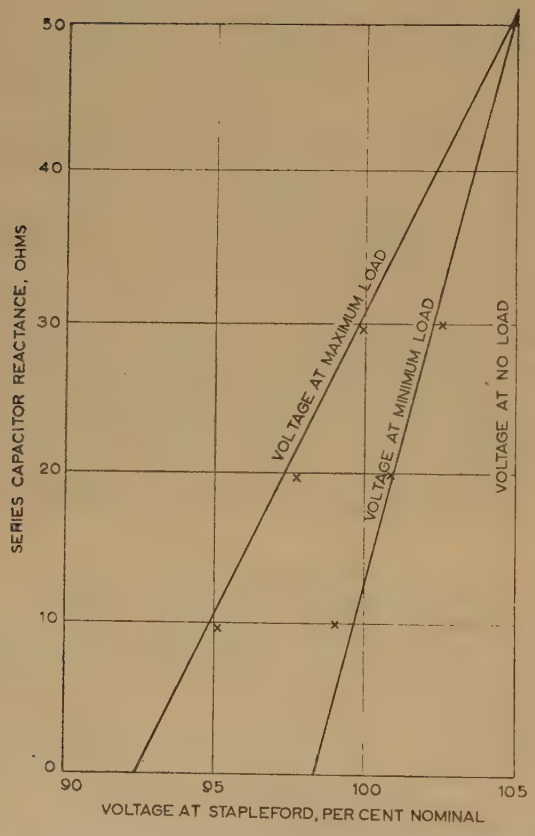


Fig. 13.—Observed performance of Penhalonga series capacitor installation with 10, 20 and 30 ohms compensation.



Fig. 14.—Skeleton map showing existing 88 kV and proposed 330 kV lines in Southern Rhodesia.

— — Proposed 330 kV line.
····· Proposed 88 kV line.
——— Existing 88 kV line.

(6.2) General Considerations

Automatic on-load tap-changing transformers are installed in the 88 kV m.s.d.s. at Norton, Mazoe and Salisbury City. It is obvious, however, that with the load growth expected these units will be operating on the highest tap position. Furthermore, there is a number of these transformers feeding into an extensive 33 kV system on which automatic on-load tap-changing transformers are installed, and these will also be operating on a high tap position; it is undesirable, therefore, with the amount of boost that the eventual load on the line will necessitate, to install further inductive methods of voltage regulation on the main 88 kV system, because voltage rise following failure of supply could cause embarrassment.

It was therefore decided to employ capacitors for regulating voltage, and a choice had to be made between switched shunt capacitors and series capacitors in order to achieve this purpose.

As the eventual load at Norton represents 2 250 000 kW-miles, β comes down to approximately 2·0 (see Fig. 7) and thus decreases the technical superiority of series capacitors. In order, however, to gain full advantage from shunt capacitors they would require to be distributed over a large number of load centres, and the cost of control gear at such load centres would be more than that for series-capacitor protective equipment. Furthermore, the shunt capacitors would be in unattended sub-stations covering an extensive area, and routine inspection of control gear would be expensive.

In view of these considerations, and the fact that the series capacitor has the characteristic feature of automatic and virtually instantaneous regulation, it was decided to employ this type of voltage-regulating equipment in this 88 kV line.

(6.3) Series-Capacitor Rating

The section of line between Umniati and Norton has a reactance of 62 ohms, but circuit reactance, including transformers, is in the region of 100 ohms. It is not desired to employ a large degree of compensation in view of the possibility, however remote, of resonance effects; on the other hand, a worth-while full-load boost is sought.

As a compromise between these factors a reactance of 50 ohms has been chosen which, at a maximum current rating of 200 amp, gives a capacitor operating voltage of 10 kV and a kilovar rating of 2000 per phase. The installation thus totals 6000 kVAr (Fig. 15). With full-load current of 193 amp at a power factor of 0·85, the line-to-line boost would be 9·4%, and would permit the automatic on-load tap-changing transformers on the system to operate efficiently.

(6.4) Siting of Capacitors

Reactance compensation between source and the first load down the line from the capacitors has to be considered. If the capacitors were to be sited on the supply side at Hartley, the degree of compensation would be high; however, since voltage-regulation difficulties at Hartley are not envisaged, it has been decided to site the capacitors on the load side.

If the capacitors were sited on the supply side at Norton the degree of compensation would be very nearly the same; but this has not been done because fault current would be higher on those occasions when, under emergency conditions, Salisbury City power station would be the source of supply with only 23 miles of line intervening.

(6.5) Capacitor Protection

Because of the low β-value prevailing, it would be costly to under-rate the capacitors to any appreciable extent. Arc-gap and contactor protection was therefore employed. It has been co-ordinated with the type of system protection envisaged, since this will have a bearing upon the desired performance of capacitor protective equipment. The line is at present fitted with inverse definite minimum-time-lag over-current and earth-fault protection; it is, however, envisaged that distance-impedance protection will be employed in the near future.

In order that the performance of distance-measuring relays shall not be disturbed by the presence of the capacitors, it has been arranged to remove the capacitors, effectively, by operation of the arc-gap during the first half-cycle of the fault. The contactors will be of the high-speed type, so that, before system breakers operate, the relays will measure distance correctly.

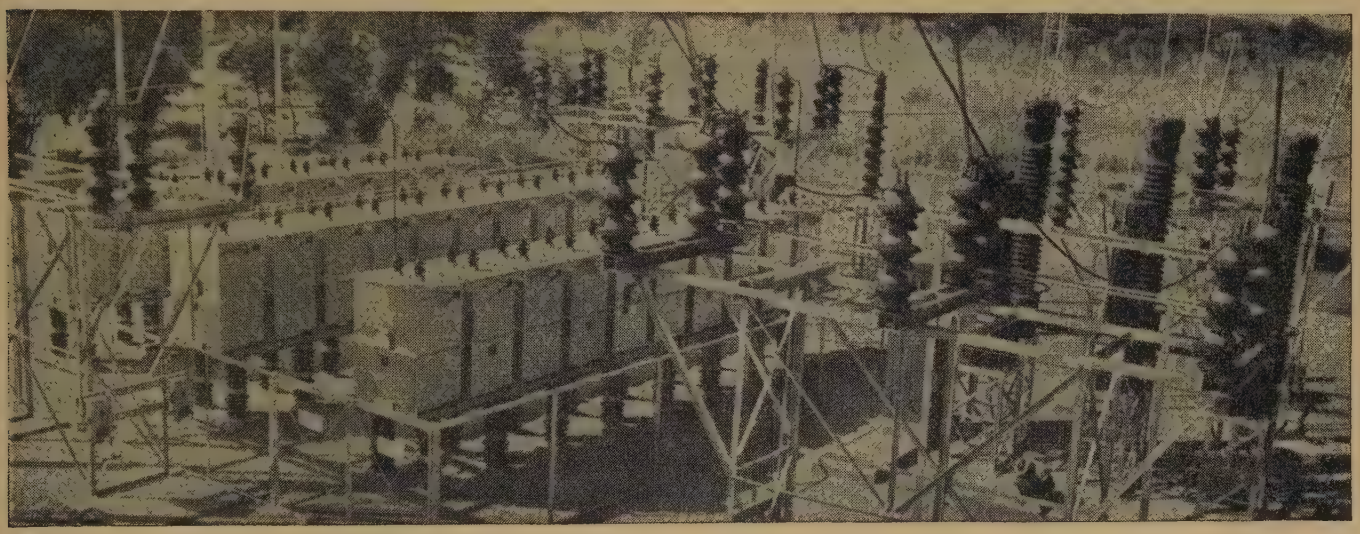


Fig. 15.—View of the series capacitor installation at Hartley.

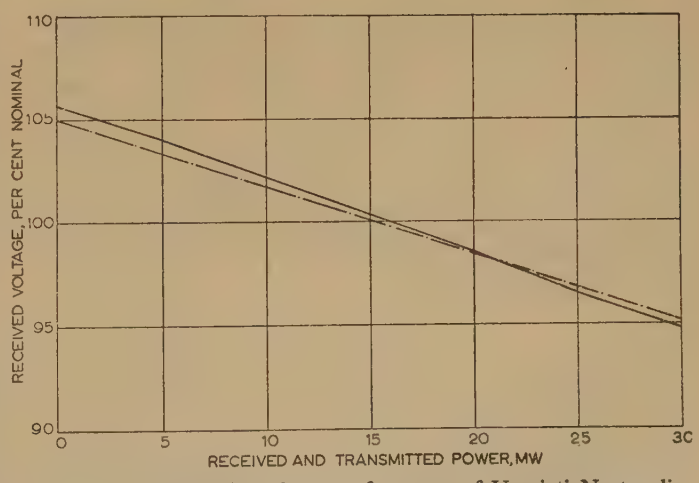


Fig. 16.—Difference in voltage performance of Umniati–Norton line with series capacitors calculated by simple and rigorous methods.

88 kV 105% nominal volts with series capacitor.
Power factor = 0·85.
——— Short solution. —— – Rigorous solution.

(6.6) Operation

The Umniati to Norton line is sufficiently long for susceptance to become significant, but not to an important extent. Fig. 16 indicates the difference between line-to-line voltage performances calculated by simple and rigorous methods.

Figs. 17 and 18 show the line-to-line voltage performance

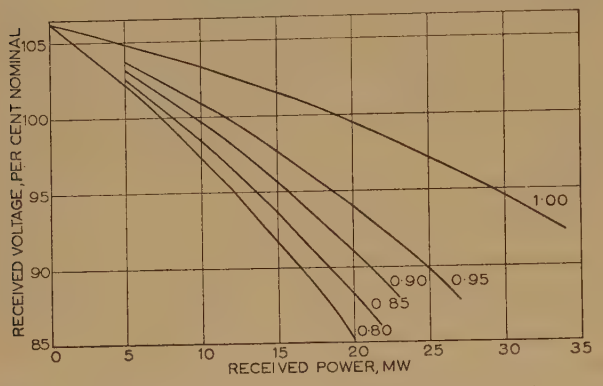


Fig. 17.—Umniati–Norton line voltage performance without series capacitors.

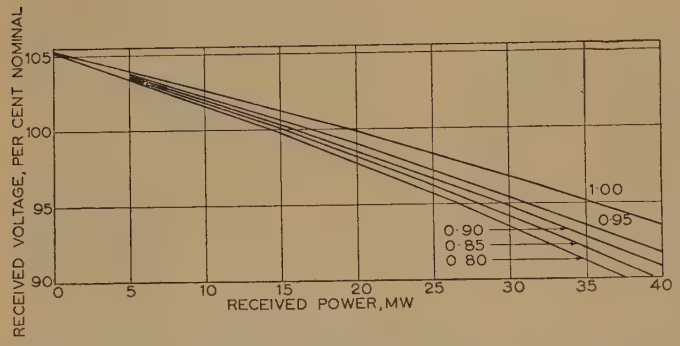


Fig. 18.—Umniati–Norton line voltage performance with series capacitors of 50 ohms per phase.

calculated by rigorous method with series-capacitor correction of 50 ohms per phase and Umniati busbar voltage maintained at 105% nominal. The effect of line charging current upon no-load receiving voltage is shown in Fig. 17, and the extent to which it is modified by the 50-ohm series capacitor is seen in Fig. 18.

At the present time, the line carries a load in the order of 10 MW, but this may be increased at any time to 28 MW as the result of a recent load exchange agreement with the City of Salisbury.

As part of the commissioning programme it was arranged that, in addition to the normal load, Umniati power station would take up part of the Salisbury City load over the line. In this manner the line load became 24 MW at a power factor of 0·91. It will be appreciated that, within the limits of acceptable voltage drop, the line would have been unable to transmit this load in the absence of the series capacitors.

Voltage- and current-recording equipment was installed at Umniati, and voltage-recording equipment at Norton and Mazoe. The calculated performance of the line with a load of

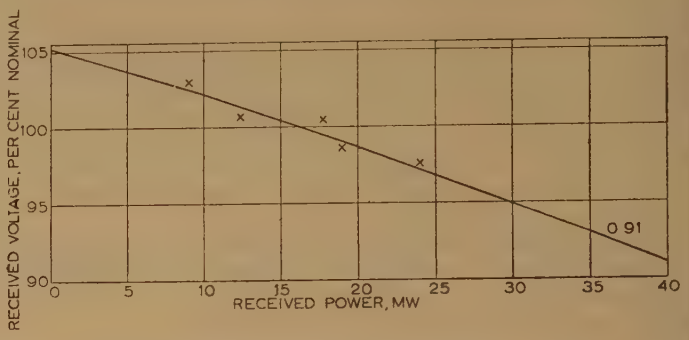


Fig. 19.—Calculated and observed performance of Hartley series capacitors with load having a power factor of 0·91.

0·91 power factor is shown in Fig. 19. Voltages actually observed at Norton are indicated on this curve and illustrate the discrepancy between calculated and observed performance.

(7) CONCLUSIONS

It would appear from the work carried out in Southern Rhodesia that the use of series capacitors provides the supply engineer with an additional means of solving his voltage-regulation problems, and is one that should not be neglected.

Series capacitors are not to be regarded as entirely replacing the older and more widely used inductive equipment for improving voltage regulation. Rather do they form a complementary technique, one which, none the less in certain circumstances, will provide the only solution to the problem.

(8) ACKNOWLEDGMENTS

The author wishes to express his thanks to the Chairman and Commissioners of the Southern Rhodesia Electricity Supply Commission for permission to publish the paper, and to members of the engineering staff for assistance given in the preparation of graphs and drawings; also to British Insulated Callender's Cables, Ltd., for their co-operation.

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(2) BUTLER, J. W., and CONCORDIA, C.: 'Analysis of Series Capacitor Application Problems', *Electrical Engineering*, 1937, **56**, p. 975.

(3) JOHNSON, A. A., MARBURY, R. E., and ARTHUR, J. M.: 'Design and Protection of 10 000 kVA Series Capacitor for 66 kV Transmission Line', *Transactions of the American I.E.E.*, 1948, **67**, p. 363.

DISCUSSION BEFORE THE SUPPLY SECTION, 22ND MAY, 1957

Mr. W. Casson: I believe this is the first paper presented to The Institution relating operating experience with series capacitors, and we are grateful for this opportunity of knowing more about this very useful tool of the transmission and distribution engineer. Although in this country we have only one installation in service at present, this does not mean that the advantages of series capacitors have not been appreciated by us. A few years ago the C.E.A. and Area Boards asked a working party to carry out an investigation and they completed a report which included the results of some full-scale field tests carried out on the system on a capacitor bank to try out various types of protective devices.

One of the conclusions given in their report was that series capacitors could be usefully employed in power systems to counteract the effect of series inductance, and could be used (*a*) to improve voltage regulation when the ratio X/R is high

and the power factor low; (*b*) to increase power transmission capacity when regulation is a limiting factor; (*c*) to improve load sharing to parallel circuits with different impedances; (*d*) to increase the stability of e.h.v. lines and by so doing increase their transmission capacity; and (*e*) to reduce flicker, for, being instantaneous in operation, they are the only means of doing this other than by reinforcing the network. Another conclusion is that the use of series capacitors is dependent on a satisfactory self-restoring protective device being available at a reasonable cost.

A study given in the supply industry's report, comparing the two methods of improving voltage regulation by series or shunt capacitors for lines of various voltages, shows that for all values of power factor for which compensation for voltage regulation may be considered, series capacitors would be more effective than shunt capacitors per kilovar in providing such compensation, but when losses are taken into account shunt capacitors may be more economic.

In Section 2.4 the author describes the advantages of over-compensation, this being particularly useful where the resistance component of the line impedance is high. In the supply industry's report a graph is given which shows the increase in transmission capacity which is possible on an 11 kV 0·1 in^2 line by the installation of series capacitors. With 100% compensation it is possible to increase the ratio of power (megawatts) to distance (miles) transmitted at 0·8 power factor from 8 to 22, and with 200% compensation from 8 to 50 without increasing the voltage regulation above 10%.

In Section 3.1 the author discusses the protection of series capacitors, which I believe is the most difficult problem associated with these devices. It is only possible to avoid providing protection if special types of capacitor units, with more insulation than that of standard types used for shunt working, are employed. Alternatively, standard types can be used if the current through them for internal and external faults does not cause over-voltages likely to result in permanent damage. A C.I.G.R.É. paper* gives some interesting information regarding the economics of increasing the insulation between foils in series capacitors, and it is concluded that a capacitor, economically dimensioned with regard to the long-time electric strength, can withstand over-voltages of 4–4·5 times the rated voltage if they are shorter than one second and occur only a few times a year. I am pleased to see that the author has one application of standard-type series capacitor on his system without protective devices which is still giving satisfactory results, but there cannot be many cases where this is possible. In the field tests carried out by the supply industry, three types of protective device were tried but none was wholly satisfactory. Two series of tests were carried out, one with no load followed by short-circuit and then the no-load condition restored, and the other with full load followed by short-circuit and then with full load restored. This latter condition was the worst with which the protective devices had to deal. The working party concluded in their report that it is unsatisfactory to have a non-self-restoring device which might short-circuit the capacitor on one phase only, leaving the two other phases in commission and thereby producing unbalance conditions.

The author makes only a slight reference at the end of Section 5.1 to lamp flicker caused by heavy currents of short duration applied intermittently; but I think that mitigation of this nuisance will become one of the important uses of series capacitors in this country, with the advent of larger arc-welding, arc-furnace and colliery-winder loads.

Finally I am pleased to announce that the supply industry in this country has agreed to the installation of several series-capacitor installations as a research project. These installations will be carefully chosen to obtain various types of experience and it is hoped, therefore, that in a few years' time we shall be able to add to the experience already being obtained from other installations, such as those described by the author.

Mr. W. S. Lovely: The paper breaks new ground in reporting on site experience at the lower values of high-voltage operation and forms an appropriate counterpart to those publications concerned with the more spectacular installations appropriate to main high-voltage transmission systems.

In the theoretical treatment in Section 2 a factor of merit, β, is used as the ratio of the rating of shunt to series capacitance for a given degree of compensation, and is used in the rest of the paper to judge the advantage of a series installation. I would have thought that a figure of merit should more properly compare a line which has series capacitance with one which has not.

The author appropriately calls attention to the fact that the inherent high resistance of light 11 kV lines damps any resonance effect that might be associated with large amounts of series-capacitor compensation.

The cost advantage of under-rating capacitors and omitting over-voltage protection across them is most interesting, and the simplicity of the installation that results is attractive for application to rural-type lines.

The maximum fault currents in the systems discussed appear to be quite low, and the growth of fault current with increased interconnection will need to be watched in relation to over-stressing of the capacitors in later years.

At the end of Section 3 the comment is made that the loss due to shunt damping resistors on the capacitors is undesirable. This is obviously true, and it is fortunate that the high resistance of the light-construction lines makes them unnecessary. With large capacitor installations on lines having a low value of β, the resistor may be necessary and the loss may have to be accepted.

From Sections 4 and 5, describing the results obtained on two 11 kV distribution feeders, it appears that these distribution lines suffered from excessive voltage drop due to the load increasing much more rapidly than had been expected, and this situation is less likely to occur in this country where compact communities are the rule. However, even if the time factor had not caused bother, the use of capacitors in the manner suggested would still be an economical way to make greater use of the rural distributors.

At the end of Section 5.3 some tests are quoted which were designed to explore the possibility of ferro-resonance. If oscillograms were taken during the motor-starting tests it would be interesting to know whether they showed any evidence of potential resonances.

Section 6, dealing with the Hartley 88 kV installation, is in a class apart. The series capacitors were installed at a time when it was expected that the 88 kV lines from Umniati to Norton would have to carry an abnormal load, and the test results show an impressive improvement in line performance. The advent of a 330 kV system may well make this problem less acute.

The capacitor protection appears to co-ordinate satisfactorily with the existing over-current protection, but it is not clear how satisfactory co-ordination is to be obtained with the proposed distance protection, unless this is slower in operation than it need be. A fault which leads to the immediate operation of the arc-gap will remove the capacitor and enable the distance relays to measure correctly, but if there are faults which do not operate the arc-gap so promptly there may be failures of co-ordination. It would be useful to have some information on the author's intentions regarding this protection.

* Nordell, R., Hogfeldt, L., and Linderholm, S.: 'Dielectric Strength of Series Capacitors', C.I.G.R.É., 1954, Paper No. 340.

Mr. R. O. M. Powell: The main obstacle to the full exploitation of series-capacitor techniques is the lack of suitable protective gear. Owing to the sensitivity of capacitor dielectrics to over-voltages, installations of even low-power banks without specific protective gear are uneconomic because of the effect of derating the capacitor units.

Until now protective gear for series-capacitor banks has, in the main, consisted of some form of spark-gap, together with an auxiliary closing contactor. The over-voltages occurring under fault conditions that the series capacitors can withstand are now reasonably well defined; thus the exact duty required of the protective devices can be specified.

The inclusion in the paper of a Section covering the siting of series capacitors on overhead lines would have been welcome, since, depending upon the degree of compensation and the location of the capacitor bank, consumers connected adjacent to the capacitor installation could experience severe over-voltages during the period when fault current flowed through the capacitor.

Mr. M. Waters: Conditions in the United Kingdom are such that there is nearly always an alternative method of improving regulation which is less expensive than the use of series capacitors. Attention has thus been focused upon the possibility of producing an inexpensive self-extinguishing protective device which would add only a fraction to the cost of the capacitor.

The self-extinguishing gap mentioned by the author extinguishes the arc and re-ignites every half-cycle of the fault current, and the flashover voltage is increased owing to the pressure of the blast of air. An improved type* is being tested in Sweden which has two air-blasts, one of high pressure, which moves the arc quickly from the spark-gap, and the other of virtually atmospheric pressure, which extinguishes the arc without appreciably raising the flashover value of the gap. The relays may be set to delay extinction until the fault is cleared, thus avoiding repeated extinctions and re-ignitions. On the same system, when compressed air cannot be used, suppression using the De-ion principle is employed. This extinguishes the arc at each half-cycle during the fault and the unpleasant voltages have to be accepted. Both devices employ a trigger gap which initiates the discharge but does not carry any fault current.

The E.R.A. has proposed an arc-chute to give self-extinction. In its simple form this, like the De-ion device, caused extinction and re-ignition every half-cycle of fault current. When the number of plates in the chute was reduced so that the fault current was not extinguished, the margin for extinction of the load current was not great. By care in design it should be possible to extinguish the load current without repeated extinctions of the fault current.

The ideal requirements of a self-extinguishing device of this type are:

(i) On flashover the arc should be rapidly transferred to arcing electrodes, where it should remain for the duration of the fault.
(ii) The arc should not be moved into the suppressing device until the fault is cleared or on the point of clearance.
(iii) A trigger gap should be used if possible.
(iv) For the United Kingdom the device must be inexpensive and require little maintenance.

In conclusion it should perhaps be mentioned that satisfactory protective equipments of the non-self-extinguishing type with short-circuiting contactors are available and have been tested in this country.

Mr. K. M. Jones: The factor β given in the paper can be misleading. It is the ratio between the shunt-condenser reactive power generation and the series-condenser reactive power generation for a given voltage boost. It gives no indication of cost comparison, because, in its derivation, no account is taken of under-rating the series condenser to cope with short-circuit currents, and transmission losses are ignored.

Both these factors can make series-condenser applications less economic than shunt-condenser applications.

The application of series condensers requires considerably more careful thought than the application of shunt condensers. For example, with regard to the location and the amount of compensation, in many cases this will depend on the permissible short-circuit currents and the effect on protective-gear operation. Compensation may have to be limited to about 40% and the condensers located near the middle of the line, in order to limit short-circuit currents for faults in the line to less than those for faults on terminal busbars, and also to avoid a non-operating zone when ordinary impedance protection is used.

Series condensers should preferably be kept away from load busbars, particularly if there are any induction motors present; otherwise a fault near the condenser could result in a change in voltage at the motors greater than normal, and consequently a current surge in excess of starting current.

The application of series condensers to high-voltage systems, for the purpose of increasing the transient stability power limit, is becoming of increasing importance. Studies of this problem have shown that it may be an advantage if the condenser is rated so that it will not flash over during the most severe transient disturbances. Fortunately, such disturbances are usually caused by faults near the sending-end generation plant, when the current carried by the condenser, being only the contribution from the receiving-end plant, may not be large. Faults near the condenser or receiving-end busbars will result in larger currents in the condenser but are less severe from the point of view of transient stability. If under these conditions the condensers flash over, an easier condition for transient stability will usually result.

Caution is needed in the application of series condensers to lines with reclosing breakers, as line currents may exceed short-circuit currents following reclosure during transient swinging conditions, and the object of the condenser in maintaining stability cannot be fulfilled if the protective gaps flash over.

With regard to damping resistors, American experience on 12 kV systems has shown that, whereas tests at minimum load indicated no need for these, voltage surges nevertheless showed up during the subsequent operation of the systems.

Mr. A. J. Haselfoot: It is useful to present voltage performance of a series capacitor in terms of active and reactive power, e.g. in the form of a circle diagram (Fig. A). It will be seen that

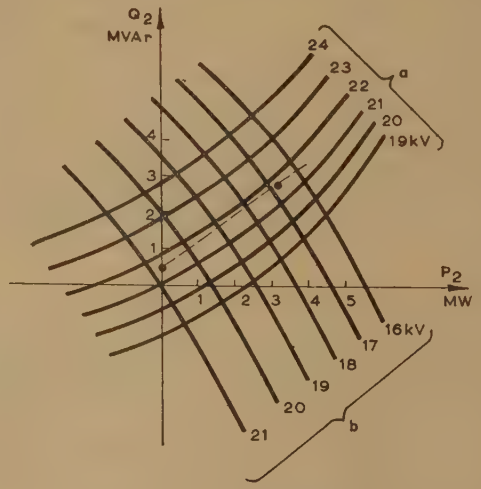


Fig. A.—Circle diagram.

* AHLGREN, L., and GRUNDMARK, B.: 'Self-extinguishing Gaps in Large Series-Capacitor Stations', C.I.G.R.E., 1956, Report No. 317.

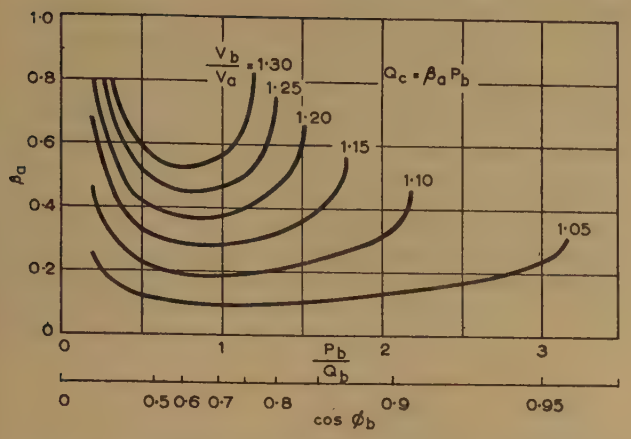


Fig. B.—Capacitor rating plotted against power factor for values of voltage rise across the capacitor.

by over-compensating to a suitable value, almost uniform voltage regulation can be achieved for a typical load curve. Regarding the rating of series capacitors, Fig. B shows curves plotted for capacitor rating against power factor for different values of voltage rise across the capacitor. It will be seen that for any given power factor there is a maximum value of the voltage rise that can be obtained which is equal to $1/\cos\phi$. As a rough and ready rule-of-thumb for average power factor of about 0·8, the increase in the line-to-line voltage is approximately equal to the voltage across the series capacitor. The ratio between shunt and series capacitance may be expressed approximately as

$$\frac{\text{Short-circuit kVA}}{\text{Load kVA}} \times \frac{\text{Reactive kVA}}{\text{Load kVA}}$$

Regarding line losses, the same rating of capacitor will give equal reduction in line losses up to the first load-point after the capacitor installation, whether this is a shunt or series arrangement. Beyond this point the series capacitor only gives voltage improvement by a reduction of current due to the voltage increase. In general, series capacitors are best for a reduction of voltage regulation and shunt capacitors for increase of power factor and reduction of line losses. A number of series capacitors have been installed in distribution lines in Sweden, and it is standard practice there to use single units of about 20–30 kVAr 1–2 kV built up to the rating required. These are used from an installation with 18 kVAr per phase on a 3-phase 3 kV line up to an installation of 35 000 kVAr per phase on a 380 kV line.

Mr. S. C. Chu: Throughout the paper I notice that the author has not mentioned the line capacitance, except at one point where he mentions that the Umniati–Norton line is sufficiently long for susceptance to become significant, but not to an important extent. Could the author tell us up to what distance of line the line capacitance becomes important to the calculations?

Concerning the location of the capacitors, I understand that with distributed loads along the line it is desirable to locate the capacitor at the centre point, i.e. at about half of the total voltage drop. This has the advantage of obtaining more compensation per ohm reactance of capacitor and also lowering the fault current. I notice in Fig. 10 that it is quite logical for the capacitor to be inserted half-way up the line, but I fail to see why, in Fig. 8, the capacitor is placed at only 1½ miles from the supply end while the total line is 40 miles long.

On the 88 kV line the distance between Hartley and Norton is probably half the length of the 90-mile line. Since the power will be transferred from either direction, I think it is perhaps wise to have the capacitor placed at Hartley instead of at Norton.

With regard to the resonance effect, the author mentions that no resonance had occurred, but there was no evidence given. I understand that in Sweden such disturbances occur quite often, and they have used a kind of protection incorporating a relay which is sensitive to sub-harmonics. This relay will short-circuit the series capacitor momentarily and sub-harmonics disappear when the capacitor is put back into the line. Perhaps the reason why resonance did not occur in the lines mentioned was the high resistance of the line or the capacity of the starting motor not being sufficiently large when compared with the total apparent power of the circuit.

THE AUTHOR'S REPLY TO THE ABOVE DISCUSSION

Mr. J. M. Magowan (*in reply*): It is gratifying to learn, from Mr. Casson, that several series-capacitor installations are to be put in commission by the British supply industry as a research project.

Mr. Casson and other speakers stress the necessity for satisfactory self-restoring protective devices, and it is in this field that most work has to be done. When such devices become available, their impact upon the performance of lines containing series capacitors may be such that many difficulties at present envisaged will lessen or entirely disappear.

It seems that protective devices for series capacitors fall into two groups: protection of the type of installation economically justified in rural lines, and protection of that similarly justified in major distribution and transmission lines. While the services performed by both would be similar, the cost of the latter could economically be 10–20 times greater than that of the former.

Apart from the difference in the current-handling capacity of the two groups of protective device, it is likely that, whereas for rural installations the device could be in operation for part of each half-cycle of fault current, with larger installations the capacitor should preferably be removed from the circuit for the entire duration of the fault but be reinserted immediately thereafter.

Mr. Waters mentions the E.R.A. requirement that for use in the United Kingdom the protective device should be cheap and require little maintenance; but these features are even more essential in colonial territories.

With satisfactory and economical protective devices it would no longer be necessary to under-rate series capacitors in order for them to accommodate fault currents. It may also prove possible to dispense with resistors shunting capacitors in order to prevent sub-synchronous resonance, even in lines of relatively low resistance.

Such resonances are due to infrequent events of short duration, such as the starting of very large motors or the excitation of unloaded transformers. The voltage developed across series capacitors by such resonant currents of low frequency would cause the protective device to remove them until normal operating conditions prevailed.

The question of co-ordination between capacitor and line protection raised by Messrs. Lovely and Jones is very important. Also important is the fact that, in order to avoid non-operating zones with distance impedance protection, the degree of compensation of a given section cannot exceed or even approach 50%.

These problems are at present being investigated on the Electricity Supply Commission's system in Southern Rhodesia and, while certain conclusions are indicated, it is not possible to anticipate them here.

Regarding over-voltage under fault conditions at consumers' premises located adjacent to series capacitors, mentioned by Mr. Powell, in some installations it could become a major factor in deciding both the degree of compensation and the location of the series capacitors. It is necessary to ensure that the product of fault current and capacitor reactance does not exceed that of normal line current and load impedance 'viewed' from the capacitors. This difficulty is, however, not likely to arise in the case of series-capacitor installations in rural lines.

Various speakers criticize the factor β because it was thought to show shunt capacitors in an unfavourable light, in that they reduce line current and loss. Actually, as Mr. Haselfoot indicates, line current and loss down to the first load centre are reduced to the same extent, whether capacitors of given rating are in shunt or series.

The significance of line susceptance, mentioned by Mr. Chu, is not primarily determined by the use of series capacitors. In general, it is recognized to be significant in lines more than about 100 miles long. In fact, the use of series capacitors tends to extend that distance somewhat.

Another aspect raised by Mr. Chu concerns the location of series capacitors at the mid-point of the line. This is of benefit only in lines which are so long that the standing-wave effect gives more compensation per kilovar of capacitor rating. In shorter lines the location of the series-capacitor installation is not very critical.

DISCUSSION ON 'ELECTRICITY IN MODERN COMMERCIAL HORTICULTURE'*

NORTH STAFFORDSHIRE SUB-CENTRE, AT STOKE-ON-TRENT, 6TH MAY, 1957

Mr. S. Scholefield: I am pleased that the authors give the heating load serious consideration, as there has probably been too much wishful thinking on low-grade space heating, such as is required in many industries in addition to horticulture.

It is disappointing to find that they so strongly favour oil and solid-fuel heating, but since the date of the paper we have had the Suez crisis, and the price of oil has altered to its own disadvantage. I think it can be said that, for the amateur and the small commercial grower, electricity for heating purposes is worth close investigation, particularly from the point of view of using off-peak storage heating. The real cost of heating at present-day prices in this area is as follows:

Fuel	Calorific value		Cost per ton	Cost per therm	Efficiency of conversion
	B.Th.U./lb	therms/ton	shillings	pence	%
Coke breeze	9 000	202	75	4·5	50
Coke	11 500	258	135	6·3	50
Average oil	19 000	425	336	9·4	60
Electricity at 1d. per kWh ..				29·3	
Electricity at ¾d. per kWh ..				22·0	

With coke breeze and coke, labour is a large consideration, and such heating would be used only where this can be kept within bounds. Oil, however, is a fairly flexible fuel, and a good installation can become almost as automatic as an electrical one. It will be noted that the actual running cost per therm after allowing for conversion is of the order of 16d., and for off-peak electricity, under 22d. This is encouraging, as previously the difference was much greater.

There is now a discernible tendency for the running cost of electricity to become lower *vis-à-vis* other fuels, and if this continues the next 10 years may produce a much more favourable state.

I think that higher soil-warming loads can be obtained, but the whole question requires much closer attention, at both national and local levels. At the Electricity Boards we are completely out of touch with many small horticultural and allotment associations, and much contact work is required here. Probably there have been many disappointments with soil-heating installations, owing to lack of water. Watering must be closely studied because it is so easy to get plant roots dried off, and after a flourishing start plants droop and die. An understanding of the larger quantities of water that are required to keep the soil moist, especially round the roots, is needed.

I agree that l.v. galvanized wire is bound to provide the safest installation, but modern p.v.c. mains cables are being used increasingly. On mains installations where the Electricity Board is consulted, the fitting of an earth-leakage circuit-breaker, where necessary, is recommended. Have the authors considered using mains cable at a sufficient depth to interpose an earthed mesh screen for mechanical protection?

With proper loadings, I think that soil warming can be suitably covered by the standard off-peak tariff.

It must be remembered that 80% practical success in the field is more desirable than perfection in the laboratory, and many unsatisfactory results have been obtained; where proper advice and supervision has been given, however, good results have been obtained, with a reduction in solid-fuel consumption owing to lower air temperatures permissible and the shorter pre-warming period required for seed beds.

I understand that trials are being made at the M.E.B. Experimental Station at Wombourne on the de-frosting of sports grounds. I recall that some de-frosting took place before the war at one of the Liverpool football grounds, under the guidance of the late Mr. P. Robinson.

I agree with the authors about heating and hope that the auxiliary units will continue to grow, but I think that the question of direct electric heating will be a most important subject in a few years' time, especially as and when off-peak usage can be arranged.

Have the authors considered the use of a heater fan working

* Cameron Brown, C. A., and Gray, A. W.: Paper No. 2272 U, January, 1954 (see 104 A, p. 249).

in conjunction with a dual-purpose thermostat for the control of high and low temperatures in small houses loaded up to, say, 6 kW?

Table 1 is an excellent example of the costs of low-grade heating which recur over and over again in many industries. It could be wished that design for heat conservation were compulsory and free from financial considerations in planning initial buildings.

Much preparatory work is required on the subject of lighting. Whether a plant is better for working overtime in its youth is an important question. Several other considerations should be studied in conjunction with this to get optimum results.

The M.E.B. has produced a plant-pot washing machine manufactured locally, and will be pleased to supply particulars.

I notice that the authors do not mention hydroponics.

Mr. J. Wainwright: The use of electrical apparatus under what are virtually outdoor conditions, with the possibilities of poor earth connections, damp atmospheres and operatives unused to electricity, could well lead to an increase in the accident rate. I would therefore be grateful if the authors could give some information on the number of accidents (fatal and otherwise) which can be attributed to the use of electricity in the horticultural industry.

One factor which could have an appreciable effect on the possibility of accidents is the care with which the electrical system is maintained. (It is assumed that in all cases the original installation will have been approved by the local Electricity Board.) Does the Authority make any recommendations to users regarding the installation of equipment, particularly with regard to suitable protective devices and methods of earthing?

The automatic operation of irrigation schemes for glasshouse and other crops has been mentioned, but no details have been given. Is it possible to control the pumping system by means of moisture-sensing probes mounted at several points in the soil? There should be little difficulty in making a fairly cheap 'moisture-in-soil' instrument which could be arranged to operate a pump delivering a controlled quantity of water. This would probably be of more interest to the amateur grower, who cannot always manage to water his crops at what is often the critical time, than to the professional who always has operatives on hand.

Mr. G. E. Castellan: I would have liked more information on installation practice for greenhouse heating, in particular as to methods of execution. I am proposing to erect a greenhouse 20 ft × 10 ft and am thinking of installing tubular heaters. As a 3-phase supply is available, I propose to make full use of it.

To this end I shall need a triple-pole contactor to control the heaters in addition to the usual thermostat and control switchgear. In view of the fact that there will be water and 415 volts in the neighbourhood, it seems to me that screwed galvanized conduit and flexible metallic connections to the heaters should be used. Surface-type p.v.c. in twins would be run in this conduit instead of the ordinary conduit variety.

If plain surface-type p.v.c. were used without conduit there might be a risk of water entering the terminal boxes of the heaters. It might be possible to bung the holes in these boxes with plastic compound, but would that be a satisfactory permanent solution? The authors' further views on this point would be of interest.

Messrs. C. A. Cameron Brown and **A. W. Gray** (*in reply*): Mr. Scholefield expresses disappointment at our acceptance of oil and solid fuel as being a practicable means of providing large-scale horticultural space heating, although his own figures and subsequent comment would seem in the end to support our factual report on the state of affairs which exists. We do stress in the paper the cogent fact that it is only by electricity, and by a very substantial use of electricity, that either of these fuels can by put to really efficient use. We do, of course, agree with Mr. Scholefield that for the amateur, and indeed for the smaller and highly specialized grower, electrical space heating has much to commend it.

We cannot, however, accept the possibility of using electricity at off-peak periods alone for direct electrical space heating in glasshouses. There is, of course, the possibility of using off-peak electricity as a boost to the other systems, but restricted electrical heating on its own would not be a practicable proposition horticulturally. There would appear to be possibilities in the storage method now being tried out by the M.E.B. at Wombourne, but at the moment we would suggest that this is more likely to benefit the amateur than the commercial grower.

With regard to soil warming, we agree that much more could be done in the way of building up this load, but experience has shown that the greatest benefit to the grower can be obtained where soil warming is used on propagating benches and in frames. This load is increasing steadily and will receive additional impetus through the greater use of electrically controlled mist propagation, which at all times should be applied in conjunction with soil-warmed beds.

Mesh screen as mechanical protection for mains-voltage soil-warming cables has been used in the past and we have had some experience with it. It introduces such serious mechanical complications in cultivation and *in situ* steaming, however, that its use has been abandoned. Soil warming is better provided through stout low-voltage wires in series-parallel grids so that ordinary cultivations can be carried out without difficulty.

Mr. Wainwright raises a most relevant point on the question of suitable installations and adequate earthing. While it is essential that any installation should be properly earthed, it is particularly important that all precautions should be taken for those in glasshouses and frames on horticultural holdings. We feel that we cannot make any recommendations better than those contained in the Wiring Regulations of The Institution, which cover the condition of the installation when connected, although we feel it is essential that inspection and maintenance should be carried out at fairly frequent and regular intervals.

We cannot give information on the number of accidents, fatal and otherwise, which could be attributed to the use of electricity in the horticultural industry, but as far as we know these are, fortunately, few and far between and certainly not to be compared with the number of accidents due to people falling downstairs.

The automatic operation of watering in glasshouses is receiving considerable attention and a big step forward has been taken in the electrically controlled mist for propagation on soil-warmed benches. The question of automatic watering is, however, a difficult one, and despite many efforts the making of a 'moisture-in-soil' instrument, which has engaged the attention of eminent people for a number of years, is by no means simple and as far as we know has not yet been successfully achieved. We agree that once a suitable and reliable instrument can be provided a big step forward would be taken towards automatic watering.

We are surprised at Mr. Castellan's suggestion that conduit is the only suitable wiring in greenhouses. P.V.C. and farm-wiring-type cable are used in the majority of installations and are proving very satisfactory; nor can the m.s.m.i. system be regarded as other than satisfactory. Indeed, we would regard steel conduit, with the difficulty of protection against corrosion, as being far from satisfactory. Provided that the right type of fittings are used there should be no trouble from watering, and provided that the correct type of tubular heaters are used there should be no necessity to bung the holes with plastic compound.

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The Institution of Electrical Engineers
Paper No. 2455 U
Dec. 1957
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THE WORLD'S COPPER RESOURCES

By H. J. MILLER.

(ABSTRACT *of a Lecture delivered before the* UTILIZATION SECTION, *9th May*, 1957.)

Introduction

Fear about the world's supplies of copper has from time to time received much publicity, and anxiety has thus been created amongst consumers, so that a statement on the facts has been considered appropriate for this Institution to have before it. Certainly the electrical industry—which is the largest consumer of copper—should not be hampered in development projects by what would appear to be unfounded anticipation of a famine at some time in the future.

It is interesting to note that for over 50 years there have been prominent subscribers to the view that a famine in copper was forthcoming. Thus Weed,[1] who was geologist to the U.S. Geological Survey, wrote in 1907 that 'The old sources of supply, though not exhausted, are getting nearer and nearer to extinction and reaching more or less rapidly the point of exhaustion'. Weed proceeded to chronicle the trends which had then recently occurred among the world's older producing mines; he reported a decreased production of copper in four of the five continents, and could present a hopeful picture only for certain States in North America.

Views of this type have from time to time been pronounced by others, and with frequent repetition the position of forthcoming shortage has come to be accepted, particularly in periods of temporary difficulties of metal supply, when sudden bursts in the rate of industrial activity cannot be accompanied by equal rates of increase of primary metals, not only copper but others as well.

An accurate presentation of the position of copper reserves was given to the Sixteenth Geological Congress in 1935:[2] it was stated that known world reserves were 100 million tons of copper, sufficient at the consumption rates then ruling to last for 69 years. Reserves may be defined as mineral of which there is tangible knowledge, through underground development, drilling or geological deduction, and to which can be attached a definite economic value. Matters of overriding importance are the cost of extraction and the selling price. Every change in these affects the ore reserves, for it is obvious that if the cost of extraction exceeds the realizable price the deposit concerned has no value. Although some mining companies publish data on their ore reserves, it must not be overlooked that these are proved amounts, and that usually the unproved deposits in operating mines are immensely greater.

Copper ore deposits consist of copper minerals—many scores are recognized by geologists, but may be classified as native copper, the oxide and the sulphide groups—disseminated in a mass of other minerals which are for the most part valueless. The copper content of most ores ranges from 0·5% to 6%, and frequently other valuable metallic minerals exist in the ores in such quantities that recovery is justified; examples are cobalt, gold, lead, molybdenum, nickel, silver and zinc.

Ore deposits extend in size from a few thousand to many millions of tons, and they may be near the earth's crust or lie at considerable depths. The low-grade deposits are massive and are mostly near the surface, where they can be exploited by open-cast or alternatively by caving methods. Vein and lobe deposits, which are usually rich and may be of isolated extent, are mostly worked by underground processes. The question of exploitation of a particular deposit depends on the complex economics of mining, transportation and extraction, and quite naturally much depends on the location.

The production of newly mined copper is supplemented by large quantities which are recovered from scrap. This is a steadily increasing source of supply, owing its origin to the fact that the uses of copper are mainly non-destructive, so that on completion of the normal life of fabricated products it is possible to make a very large recovery of the copper content. Reserves of this character are continually growing, and if there should occur a long period in which there was stability of applications, the supply of copper derived from scrap sources could exceed virgin metal supplies. However, the applications of copper are steadily expanding and primary metal production is also increasing, so that the maximum impact of scrap copper as a source of supply has not been experienced. The pressure which can be exerted by scrap supplies will be recollected by many who have experienced the ruthless effects which supplies of scrap have on the market in periods of depression.

Primary Copper Production

Statistics of primary metal production, as published by the American Bureau of Metal Statistics,[3] are summarized in Table 1, giving the production by 5-year periods since 1916. This shows that production has been subject to many abnormal influences, especially in the last 25 years, owing to the considerable variation in demand during the slump of 1930–35, the war period and the post-war boom. During the latter, demand has often greatly outstripped supply, in spite of the major expansion schemes of Rhodesia, Canada, the Belgian Congo and other countries. Here it must be remembered that mine development and expansion occupy periods of 2 to 5 years; hence the impossibility of matching supplies with sudden upturns in demands.

The United States is both the largest producer and the largest consumer of copper, at present taking almost one-half of the world's primary production. The overall reserve situation was fully considered by a recent Commission,[4] in whose report—known as the Paley Report—it was estimated that the available U.S. ore reserves contained 25 million tons of recoverable copper, and that the justifiable production rate of copper for the next 25 years would be 800 000 tons a year. It was considered that new reserves, which would be brought in either as extensions of known deposits or as new occurrences, would approximately match this production rate, but any greater amount of extraction would result in a depletion of reserves. Thus for the foreseeable future, the United States would find it necessary to import copper on a gradually increasing scale, so as to meet the expected increase in industrial demand, currently running at about 1·4 million tons of primary metal. This position contrasts with the situation some 20 years ago when the United States was an exporter of copper and the urgent need was for U.S. mining companies to invest in the development of oversea properties.

The backbone of U.S. copper production is from the massive

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Table 1

WORLD PRODUCTION OF COPPER, 1916 TO 1955, IN 5-YEAR PERIODS: SHORT TONS

	1916–20	1921–25	1926–30	1931–35	1936–40	1941–45	1946–50	1951–55
	×10³	×10³	×10³	×10³	×10³	×10³	×10³	×10³
U.S.A.	4141	3171	4389	1491	3634	5006	4001	4675
Canada	246	194	503	809	1401	1422	1188	1409
Mexico	313	220	360	237	232	280	337	314
Peru	226	203	271	165	197	188	139	198
Chile	501	829	1398	1117	1888	2666	2168	2149
Other American countries	90	112	117	54	117	95	123	117
Europe (excluding U.S.S.R.)	379	509	645	717	787	547	579	605
U.S.S.R.	41	18	127	222	419	750	1006	1663
Asia	492	337	432	466	664	553	394	750
Australasia	181	83	70	82	112	132	83	166
Belgian Congo	*	*	*	504	707	894	846	1180
Rhodesia	*	*	*	540	1156	1277	1286	1960
Other African countries	193	414	725	65	89	128	200	294
	6803	6090	9037	6469	11403	13938	12350	15480

* Included in the figures quoted for 'Other African countries'.

deposits of porphyry coppers, having a low average copper content. These deposits are of enormous size, particularly with respect to horizontal dimensions, ranging from 600 ft × 2000 ft to 4000 ft × 6000 ft, while their depth is also considerable, ranging from 400 ft to 2000 ft. The copper minerals are disseminated more or less uniformly throughout the mass, mostly running at about 0·8% to 1·5% copper, and are present as sulphides, except near the surface where oxidation has occurred, so that the copper has in some instances been converted to the oxidized form and in others has been leached out leaving a barren cap.

Northern Rhodesia has undergone rapid expansion since 1931–32, when Roan Antelope and Rhokana first went into production, and there are now four large and two small producing mines, with development plans in hand at other properties. Difficulties of transportation and supplies were the foremost factors delaying the commencement of operations, and these factors have restricted the expansion rate. Indeed, it has frequently been necessary to adopt costly expedients in order to overcome various obstacles and thus maintain production. As time has progressed many of the shortcomings in transport, fuel supplies and power have been overcome, and further large-scale developments in these services are at present under way.

While production has increased from negligible tonnages in 1931 to about 350000 tons in 1955, there are production plans in hand for increasing this to about 600000 tons in 1960–65. The ore deposits at all the properties are comparatively rich, running from about 3% to 5% copper, and the companies' published ore reserves are about 700 million tons; these reserves are increasing year by year as the properties are developed. Exploration work is also being undertaken over a much wider area than the present copperbelt boundaries, in the fullest confidence that other valuable deposits will be revealed.

In the adjacent Belgian Congo, the Union Minière du Haut Katanga has expanded throughout the 46 years of its existence, in which period it has produced almost 5 million tons of copper. It is currently yielding over 250000 tons a year from 12 mines located across the 200-mile concession,, and of these many are new and rich deposits, averaging over 6% copper, and frequently carrying other metals. The company has not published data on ore reserves, but from their recent investments in mining, smelting and refining plants it can confidently be expected that these are considerable; in fact the Paley Report estimated the copper deposits of the Congo to contain 40–80 million tons of metal.

Chile has three immense mines, which have yielded about 400000 tons a year of copper for upwards of the last 20 years. While the properties possess great natural advantages, so that direct operating costs are low, these are countered by the excessive taxation and discriminating exchange rates imposed by the Chilean Government. There have also been other unfavourable factors, and on the whole there has been created within recent years a condition of uncertainty, in which the wisdom of further investment has even been questioned. In spite of this, the American owners have undertaken several immense schemes which have improved production capacity, so that, given reasonable stability, there should be a great improvement in output record. For instance, it has been forecast that in 1960 an output of 600000 tons a year should be obtained.

Peru has been a constant producer of copper for 400 years, but most of the small mines working on outcrops of rich ores have abandoned operations, so that in recent times the mainstay of the Peruvian production of 30000 tons a year has been the Cerro de Pasco mine. However, great activities are currently being taken in regard to several extensive deposits of low-grade ore in the southern part of Peru. Operations are now in an advanced stage for the production of 140000 tons a year of copper by the American Smelting and Refining Co., in association with other companies, from the Toquepala deposit. This is a massive porphyry deposit with over 1% copper, which has not yet been fully assessed, though the reserves already proved are in excess of 400 million tons. The open pit is being prepared and smelting plant erected for production of blister copper to commence in 1958–60. Other known deposits of the porphyry type are being investigated, and it appears that many of these will also be developed so that in the none too distant future there should be a great increase in output from Peru.

Copper and other metal mining activities in Canada have been described as still in the pioneer stages, and it is noteworthy that scarcely a month goes by without a report of the discovery of a new deposit or of plans for increased production from an existing mine. The copper production industry comprises four large groups—International Nickel, Noranda, Hudson Bay and Falonbridge—but in addition there are many small independent companies. While several deposits are of immense size, most of

them are small and comparatively rich, so that they lend themselves to profitable working on a small scale. It is a feature of Canadian orebodies that practically all of them contain valuable amounts of other metals, notably nickel, lead, zinc, silver and gold. Hence copper production in many cases is incidental to that of other metal recovery, so that it is partly dependent on sales and policy matters associated with the other metals.

In many other countries, copper production has made great headway, this being particularly the case in Russia, whose production, now about 400 000 tons a year, is expanding rapidly.

Secondary Copper Production

Copper and copper-alloy scrap is classified into two important groups, 'old scrap' and 'new scrap', which together comprise secondary copper supplies. However, from the point of view of supplementing primary metal supplies, it is only the old scrap which counts.

This arises from the scrapping of miscellaneous equipment, plant and structures, and comes to the market via dealers as and when it arises. The quantities are unpredictable, and yet they are not so unstable as might be thought. Statistics of old scrap supplies are not readily available, except in the United States, where the Bureau of Mines[5] has compiled the necessary information, which indicates as a round figure 450 000 tons a year. Similar data for other countries are not available, though it might be deduced from the overall figures available that old scrap makes a generous contribution to the total. Though it is dangerous to generalize, it is possible that the contribution of old scrap amounts to 20–30% of the primary metal consumption, which indicates a world total scrap at present exceeding a million tons a year.

Amount of Copper in Industry

The sum total of the production of primary copper from 1800 to 1955 is almost exactly 100 million long tons, and the 'pool' of metal which industry is at present employing will only be a little short of this. This 'pool' is at present being added to at a rate close to 3 million tons a year, this being the primary metal production.

Assuming the reclamation of old scrap throughout the world at a million tons a year at the present time, it can thus be accepted that there is available 4 million tons a year of metal. It is quite impossible to state the extent of the wastage which occurs in use, and hence the potential degree of salvage which can be achieved, but it is considered that there is scope for a greater reclamation than is at present attained. It may, for instance, be noted that the Paley Report[4] looks to a recovery of 700 000 short tons a year in the United States by 1975; on this basis the world's secondary copper recovery might well be 1½ million tons a year.

Backed by the immense proved reserves for primary metal production—estimated at a total equivalent to 50 years' requirements, with the prospects that there will be little if any diminution of these reserves as the years go by—and by the hundredfold cover for the secondary metal production, it can with confidence be claimed that the overall supply position for copper is satisfactory. However, it is inevitable that from time to time production and consumption should get slightly out of step, particularly as primary metal producers can obtain increased production only as a result of long-term plans.

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(4) 'Resources for Freedom' (U.S. Government Printing Office, 1952), Volume 2, Chapter 5.
(5) 'Minerals Yearbook' (U.S. Bureau of Mines, Washington, D.C.).

DISCUSSION ON 'POWER SYSTEM PROTECTION, WITH PARTICULAR REFERENCE TO THE APPLICATION OF JUNCTION TRANSISTORS TO DISTANCE RELAYS'* AND 'A DUAL-COMPARATOR MHO-TYPE DISTANCE RELAY UTILIZING TRANSISTORS'†

NORTH-EASTERN RADIO AND MEASUREMENT GROUP, AT NEWCASTLE UPON TYNE, 18TH MARCH, 1957

Mr. F. H. Birch: In Paper No. 2085 S the authors point out that, by applying carrier protective equipment with transistors, the complex power supplies now required for such equipment using thermionic valves could be dispensed with. Before this is done, however, I hope that due attention will be paid to the battery supplies already available at high-voltage substations, as an equipment capable of using such supplies is much to be preferred to an equipment requiring either a separate battery, charger and monitoring equipment or a motor-generator and associated control gear.

The 275 and 132 kV stations in this country are all provided with 110-volt batteries for switchgear tripping purposes. These batteries and their associated wiring are normally unearthed to minimize the risk of insulation faults leading to unwanted tripping. All wiring and apparatus associated with tripping batteries must be capable of withstanding a test of 2 kV to earth.

The 275 and 132 kV stations are also equipped with 50-volt

* ADAMSON, C., and WEDEPOHL, L. M.: Paper No. 2085 S, August, 1956 (see 103 A, p. 379).
† ADAMSON, C., and WEDEPOHL, L. M.: Paper No. 2177 S, September, 1956 (see 103 A, p. 509).

batteries for telecommunication purposes. These are, with a few exceptions, earthed at the positive pole, and all wiring and apparatus connected to them should be capable of withstanding a test of 250 volts d.c. to earth for one minute.

It would not appear economic to apply carrier protection to feeders working at voltages below 132 kV.

Mr. E. J. M. Marrian: I agree with the authors that existing types of electromagnetic techniques are satisfactory for the large majority of requirements. The greater speeds obtained with relays of the electronic type have their application where system-stability problems exist, but in other cases I feel that operating reliability and accuracy of measurement should not be sacrificed for speed of operation.

Truly high-speed protective relays are affected by transient conditions, not only in the system but also in the current transformers themselves. These latter transients may be troublesome, and I should be glad to know whether the authors have found any method of combating them.

It would be interesting to know of the authors' experience of transistors themselves. Has there been any tendency to deterioration during the experiments, and, if so, is it related in any way to the total energized time? In other words, do they retain their properties if left de-energized for any length of time?

The best way of obtaining experience with any type of protection is to make a prototype and install it in an actual transmission system so that it may operate as in service and be subject to all the hazards which protective gear normally has to undergo.

I think the accuracy/range charts shown in Figs. 11 and 12 of Paper No. 2085 S give a most useful method of comparing protective-gear capabilities, and I hope that this method may be more widely applied.

Are the characteristics shown in Fig. 16 drawn from actual test results? If so, the authors have produced an excellent mho-type relay, and I hope their difficulty with regard to transient overreach will soon be solved.

***Mr. G. F. L. Dixon:** On distribution systems (I am thinking particularly of 33 kV overhead-line systems) lines tend to become shorter and earth-fault currents are often small. Sooner or later one finds oneself trying to protect lines a few miles long, traversed by earth-fault currents of 250 amp or less. For these reasons I believe that any distance relay chosen as standard for a distribution system should have a useful range-factor of at least 40. Until now, nothing but an electromagnetic reactance relay has been able to achieve such a performance. In particular, electromagnetic admittance and impedance relays have not.

The authors have now developed a transistor-comparator admittance relay which has a useful range-factor rivalling that of the electromagnetic reactance relay. I think that this is one of the most striking features of the new relay. In view of the advantages of admittance relays as such (simplicity, marked immunity from power swings and the ability to protect very long lines), is the death knell sounding for the reactance principle? I think not. Apart from the question of range factor, the reactance relay has another important advantage. Its accuracy is substantially unaffected by arc resistance and earth-fault contact resistance. Both these upsetting influences are likely to grow more important as we build more unearthed lines, and as the distances between relay points become shorter.

* This contribution was unavoidably held over from the discussion before the Mersey and North Wales Centre, at Liverpool, 18th February, 1957.

As a separate issue, I should like to ask the authors a fundamental design question. In the same way that an electromagnetic reactance relay has a higher useful range factor than an electromagnetic admittance relay, would a transistor-comparator reactance relay, if developed, have a higher useful range factor than a transistor-comparator admittance relay?

Messrs. C. Adamson and **L. M. Wedepohl** (*in reply*): In reply to Mr. Dixon's question on the relative range factors of transistor-type distance relays, there is generally very little variation from one relay type to another. However, there is very little difficulty in increasing the range factors considerably, values of 100 to 120 being physically realizable. The increase in burden is not significant since the initial burden is so low.

We do not think that the mho-type relay will oust all other types. This class of relay is, in fact, most suitable for long overhead lines and tends to lose accuracy on the shorter lines where arc resistance becomes appreciable. The reason for concentrating on the mho-type characteristic in the papers was that these units give most information on both distance measuring and correct directional discrimination. Laboratory relays have shown that there is no difficulty in obtaining the reactance characteristic if required.

However, we do not feel that the transistor-type relay with limited earth-fault currents would be suitable for very short lines. In such cases the use of very fast relays would not be economically justified when slower schemes would be acceptable.

With reference to the remarks made by Mr. Birch, the power supplies do present a problem. The station battery could be used in one of two ways, namely to energize a voltage-limiter circuit from the battery or alternatively to trickle-charge a sealed cell or accumulator. In either case the equipment would have to be insulated to withstand a 2 kV test to earth.

In reply to Mr. Marrian, it is our view that relays operating on the direct-phase-comparison principle and using a dual-comparator configuration are free from maloperation owing to system transients or those subsequently originating in current transformers. It is not necessarily the case where high-speed relays operating on pulse principles are concerned. This has been commented on in Paper No. 2085 S and was the reason for the discontinuation of experiments on the pulse-type relays.

For the last two years a life test on 40 transistors has been carried out in the Power Systems Laboratory at Manchester; these transistors have been working under a variety of conditions. Ten of them have been working under conditions of approximately twice the normal power dissipation, and after a period of a few months, five of these changed their characteristics. It is interesting to note, however, that the transistors then became stable at their new internal impedance and that their performance as switching elements was unimpaired. None of the other 35 transistors have undergone any change in characteristic whatsoever. In our opinion good circuit design can take into account the possibility of any change in characteristic of this type, although there is no reason why any transistor should be placed in a circuit under conditions where the normal power dissipation is exceeded.

We agree that installation on an actual transmission system is desirable, and steps have been taken to arrange for a complete terminal to be put into service on a line of high lightning-fault incidence in South Africa.

With regard to Mr. Marrian's last point, the characteristics of Fig. 16 were drawn from actual test results.

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The Institution of Electrical Engineers
Paper No. 2437 M
Dec. 1957
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THE DYNAMIC PERFORMANCE OF A MAGNAVOLT CASCADE EXCITER

By R. A. STEVENS, M.Sc.(Eng.), Associate Member.

(*The paper was first received* 14*th March, in revised form* 1*st July, and in final form* 26*th August,* 1957.)

SUMMARY

The purpose of the work described was to relate the measured voltage and time-constants of the machine to the frequency and step responses. A brief description of the machine is followed by an outline of the methods adopted to measure its constants, and a statement of the technique for measuring the frequency response is followed by the results obtained. The step-response tests are next described and discussed, and finally a direct transform from step-response to frequency response is put forward.

LIST OF SYMBOLS

k_1, k_2, k = First-stage, second-stage and overall voltage constants, volts/amp.
p = Laplace transform variable.
F_2, F_4 = M.M.F. per pole, 2-pole and 4-pole field windings respectively, AT.
F = First-stage armature m.m.f., AT.
S_1–S_4 = Reluctances of gaps 1–4, respectively.
Φ_{12}, Φ_{34} = Fluxes between poles 1 and 2, 3 and 4, respectively.
Φ_2 = Two-pole field flux.
Φ = First-stage armature flux.

(1) INTRODUCTION

The rotary amplifier upon which the tests were made was a Magnavolt exciter, which, unlike the cross-field machines, has two separate armature windings and two systems of field windings, and is described as a cascade exciter. The tests were made in order to determine whether it would be possible to predict the frequency- and the step-function responses from measurements of the winding resistances and inductances and the voltage constants of the machine over its linear range.

A direct step-function to frequency-response transformation was also used to determine the frequency response from the measured step response.

The exciter used was a 2 kW 200-volt 10 amp 1500 r.p.m. unit; speed variations during the tests were small, and no allowance was made for them.

(2) DESCRIPTION OF THE MAGNAVOLT

The Magnavolt consists of two d.c. generators in one frame with separate armature windings accommodated in the same slots[1–4] (Fig. 1), the 2-pole and 4-pole windings being for the first and second stages respectively. The four main poles on the laminated field-system are excited by winding Y_1–Y_2 in Fig. 1, and induce a voltage in the 4-pole armature winding but not in the 2-pole winding. The field winding X_1–X_2 (the input winding) produces a 2-pole field system which induces a voltage in the 2-pole armature winding. This is connected to Y_1–Y_2 and excites the second stage, thus giving two stages in cascade. The second stage is provided with interpoles, which are unnecessary in the first stage since the armature current is small.

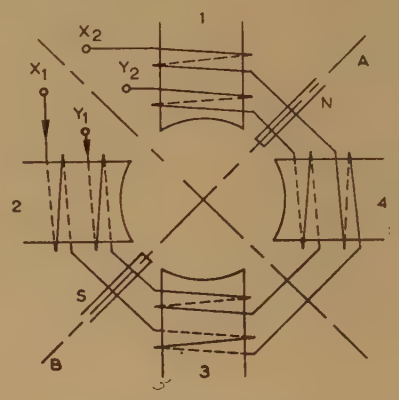


Fig. 1.—Field windings.
X_1–X_2. Two-pole field winding.
Y_1–Y_2. Four-pole field winding.

(3) DETERMINATION OF TRANSFER FUNCTION

The coefficients in the transfer function were determined by measurement under various conditions.

(3.1) Relationship between Generated E.M.F. and Field Current

Open-circuit tests gave the following constants for the machine:

		volts/amp
First stage	k_1	= 88
Second stage	k_2	= 105
Complete machine	k	= 150

Hysteresis effects are small but not negligible, and the machine is linear up to an output of about 150 volts.

(3.2) Winding Resistances

(3.2.1) Field Windings.

The resistances at an ambient temperature of 20° C are: 2-pole field-winding, 4·2 ohms, made up to 10 ohms for tests; 4-pole field-winding, 5·6 ohms.

(3.2.2) Armature Windings.

The resistances were obtained by measuring the change in the terminal voltage when the machine was loaded. The results when loaded on a separate resistor differed slightly from loading on the second-stage field in the case of the first stage, and differed widely from the standstill value of 4·5 ohms, the measured value being about 7 ohms, depending on the armature current.

The variation is due to feedback brought about by a direct component of armature reaction, the main causes of this being variations in air-gap lengths over the pole arc (see Appendix), discrepancies in brush positions, and saturation of leading pole-tips. There is also feedback from the second-stage field, owing to a non-linear iron circuit and to unbalanced air-gaps. The effective second-stage armature resistance is 4·5 ohms, there being additional feedback here because the interpoles are on the same axis as the input field.

Written contributions on papers published without being read at meetings are invited for consideration with a view to publication.
Mr. Stevens is at the Welsh College of Advanced Technology.

(3.3) Winding Inductances

(3.3.1) **Field Windings.**

Measurement at 50 c/s was found to be unsatisfactory. Direct measurement of fluxes using search coils and a Grassot fluxmeter gave an inductance of 1·0 henry for the input field winding and 1·9 henrys for the second-stage field winding. An indirect determination of the flux per pole from generated-voltage and armature-winding particulars led to values of 0·94 and 1·7 henrys respectively, which are less reliable, owing to leakage effects.

(3.3.2) **Armature Windings.**

Since the poles of the first stage are split into two projections, it is possible to measure the armature flux directly, using search coils on these projections. By this method the gap reluctance was found and hence the first-stage armature inductance was calculated,[1] the value being 0·265 henry.

(3.4) Time-Constants and Transfer Function

The input field circuit has a total resistance of 10 ohms, an inductance of 1·0 henry and a time-constant of 0·1 sec.

The inter-stage circuit has a resistance of 12·5 ohms (this is a mean value, for there was a variation as noted in Section 3.2.2), an inductance (first-stage armature plus second-stage field) of 2·3 henrys, and a time-constant of 0·18 sec.

Transfer functions:

$$\text{First-stage} \quad Y(p) = \frac{8 \cdot 8}{(1 + 0 \cdot 1p)} \qquad (1)$$

$$\text{Second-stage} \quad Y(p) = \frac{8 \cdot 4}{(1 + 0 \cdot 18p)} \qquad (2)$$

$$\text{Overall} \quad Y(p) = \frac{75}{(1 + 0 \cdot 1p)(1 + 0 \cdot 18p)} \qquad (3)$$

(4) FREQUENCY-RESPONSE TESTS

(4.1) Testing Equipment

The equipment used for the tests is similar to that described by Burns and Cooper,[5] and is shown in outline in Fig. 2. A velodyne[6] speed-control drives a synchro which has a 1 800 c/s voltage fed to its rotor winding. The stator windings thus give

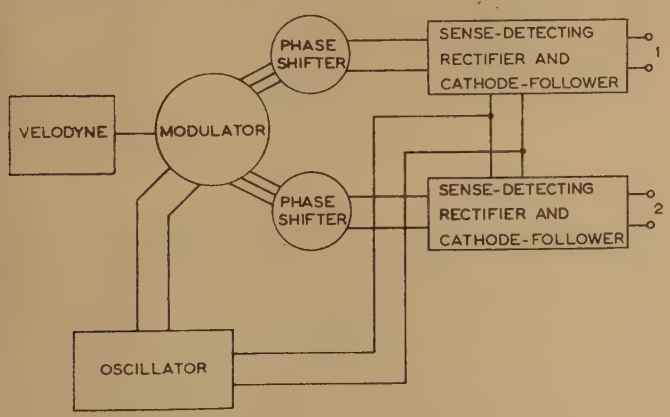


Fig. 2.—Frequency-response testing equipment.

3-phase voltages at 1 800 c/s, modulated at the rotational frequency of the synchro. This voltage is fed to the stator windings of two similar synchros in parallel which act as phase shifters. The rotor windings of these two synchros are connected to two rapid-response sense-detecting rectifiers,[7] each of which is directly coupled to a cathode-follower output stage via a potentiometer. Two outputs are obtained which are variable independently in magnitude and phase in the frequency range 0·1–50 c/s. One of these outputs is fed directly to the X-plates of a cathode-ray oscillograph with a long-persistence screen, and the other is fed via an amplifier to the Magnavolt field-windings; the output of the Magnavolt is fed via an attenuator to the Y-plates of the oscillograph. The amplifier is direct-coupled and has four 25-watt beam tetrodes in parallel in the output stage with feedback from their cathodes to the driver stage. The machine field is connected in the cathode circuit, the standing current being balanced out by a negative bias supply. Tests carried out on the amplifier showed that the output resistance was less than 1·0 ohm while the harmonic distortion was about 2% with a load resistance of 10 ohms and the amplification set to 0·2 by means of the feedback circuit.

The method of use was to close the phase ellipse on the oscillograph and read the phase shift from the phase shifter, and then to adjust the attenuator until the oscillograph deflection reached a predetermined value; the voltage amplification of the magnavolt could then be read directly from the attenuator.

(4.2) Results of Tests

Tests were made on the two stages individually and on two in cascade, at various input levels.

(4.2.1) **First Stage.**

The locus diagram in this case should be semicircular,[9,10] and a representative diagram is shown in Fig. 3. It will be seen

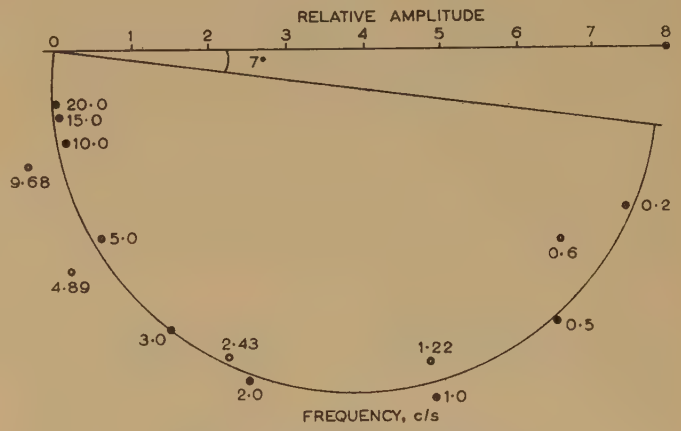


Fig. 3.—First-stage locus diagram.
● ● ● Measured frequency response.
○ ○ ○ Derived from step response.

that the diagram is almost semicircular, but that the diameter of the semicircle lies at an angle of about 7° to the horizontal axis. This is due to the additional phase shift at low frequencies caused by hysteresis,[8] which is borne out by the fact that the curve of measured phase shift, θ, plotted against arc tan ωt, on extrapolation, cuts the θ-axis at about 7°. The points for higher frequencies lie inside the semicircle, since the phase shift is reduced at higher frequencies by eddy currents in the field system—an effect which is very marked if plain shims are used to modify the gap. The loci can be predicted quite well from the measured constants, and the hysteresis angle can be estimated from the hysteresis loop.

The logarithmic graph (Fig. 4) shows that the measured response agrees very well with the asymptote drawn from the measured constants. The phase-angle curve differs from the predicted one by the angle introduced by the hysteresis effect.

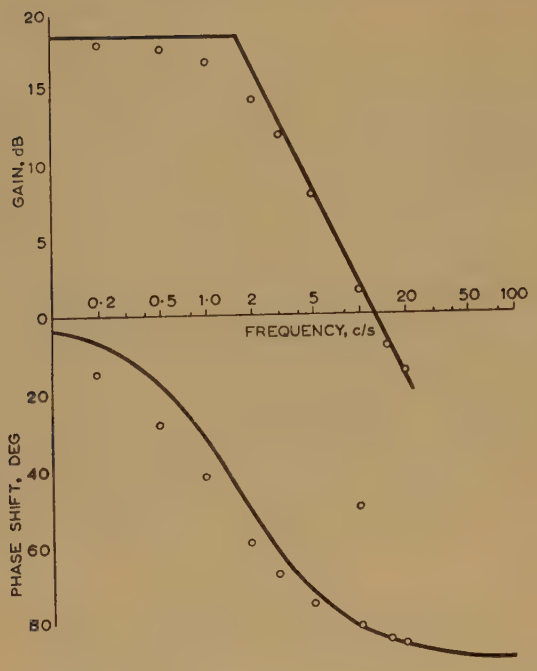


Fig. 4.—Gain and phase plotted against log-frequency for the first stage.
——— Calculated values.
○ ○ ○ Measured values.

It will be seen that there is a discrepancy in the amplification from the open-circuit and frequency-response tests; the cause of this was not discovered, although careful checks were made.

The addition of resistance to modify the time-constants altered the time scale and amplification in the manner expected.

(4.2.2) **Second Stage.**

The diagrams for the second stage are very similar to those for the first and do not require further discussion.

(4.2.3) **Two Stages in Cascade.**

The locus diagrams obtained are of the general shape expected (see Fig. 5), although, owing to the greater complexity, it is not possible to see discrepancies as it is with a single stage.

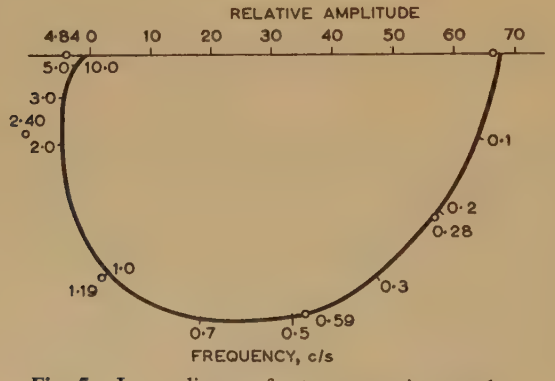


Fig. 5.—Locus diagram for two stages in cascade.
○ ○ ○ Values derived from step response.

When the machine was loaded the amplification was reduced and the phase shift increased slightly, owing to the introduction of a third circuit having phase lag. When the peak input current was increased to 300 mA the output became distorted, and it was impossible to obtain a straight line on the oscillograph, the system being insufficiently linear for accurate measurement to be made. The locus diagram obtained differed widely from one predicted from the measured constants, as would be expected.

Two logarithmic graphs are shown in Fig. 6, the lower one

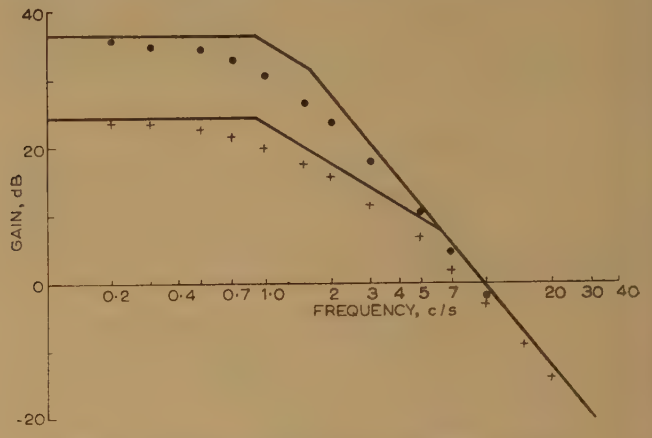


Fig. 6.—Gain/log-frequency for two stages in cascade.
——— Calculated asymptotes.
● ● ● Measured response.
+ + + Measured response with reduced first-stage time-constant.

being for added resistance in the input field circuit. The divergence from the asymptotes is rather more than would be expected theoretically, and also more than for a single stage, but the asymptotes would be sufficiently accurate for most design purposes.

(5) TRANSIENT RESPONSE

Tests were made under the same sets of conditions as for the frequency-response tests, using an input step-function of voltage.

(5.1) Method of Performing Tests

The same amplifier was used as for the frequency-response tests, but the input was a known voltage step obtained from a voltage divider via a switch. The record was made on 2-channel high-speed pen recorder with the amplifier input on one channel and the Magnavolt output on the other. The recorder had a substantially flat frequency response up to 60 c/s, and the pen length was large compared with the trace amplitude, the error due to the curvature of the trace not being corrected. The trace amplitude was about 2 cm, which was rather small for accurate measurement. The paper speed was 15 cm/sec in all cases.

(5.2) Test Results

(5.2.1) **First Stage.**

Fig. 7 shows an example of the response of the first stage, with the simple exponential curve which is expected with a single time-constant. The general agreement is quite good, but there are definite signs that there is more than one time-constant affecting the response, possibilities being a momentary slowing of the set when the step is applied and the effect of mutual coupling of the field winding with the short-circuited coil undergoing commutation. When the response was plotted on log-linear co-ordinates, a fairly good straight line resulted which gave a time-constant 10% low in the worst case, probably due to inaccuracies in measuring the trace.

(5.2.2) **Second Stage.**

The results here were very similar to those for the first stage and are not discussed further.

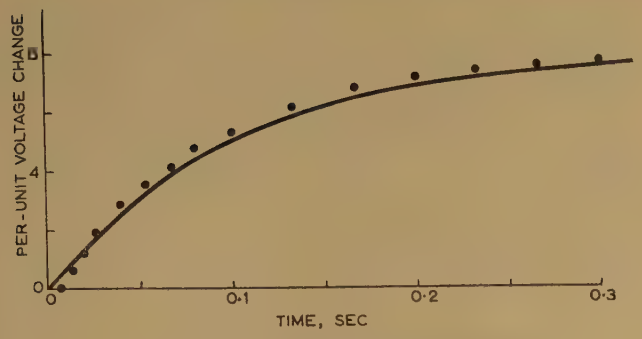


Fig. 7.—Step-function response for first stage.
——— Calculated values.
● ● ● Measured values.

(5.2.3) Two Stages in Cascade.

A typical response is shown in Fig. 8, together with the calculated response using measured values of the constants. The error in the final amplitude is mentioned in Section 4.2.1.

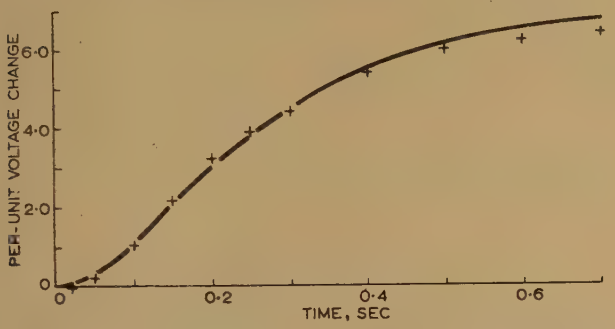


Fig. 8.—Step-function response for two stages in cascade.
——— Calculated values.
+ + + Measured values.

In one of the tests the time-constant of the first stage was reduced, by adding resistance, by a factor of 10, the time-constant ratio being 18 : 1. On plotting on log-linear co-ordinates (Fig. 9)

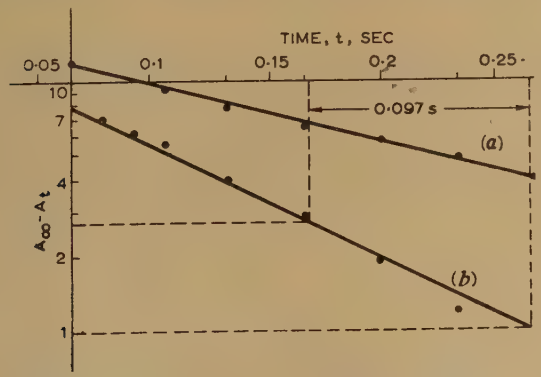


Fig. 9.—Step-response plotted logarithmically.
A_t = Amplitude at time t.
A_∞ = Amplitude at $t = \infty$.
(a) $T_2/T_1 = 18$.
(b) $T_1/T_2 = 5{\cdot}5$.

a reasonably straight line was obtained giving 0·18 sec for the second-stage time-constant. A similar test gave 0·097 sec for the first-stage time-constant, the ratio here being only 5·5 : 1. These values are sufficiently accurate for this method to be useful. The graphs are curved for small values of time, owing to the effect of the smaller time-constant, but its effect is negligible where the ratio of the time-constants is greater than about 10.

(6) STEP-FREQUENCY RESPONSE TRANSFORMATION

As indicated in Reference 12, a step-frequency response transformation can be very useful. Any mathematical transformation of this sort, based on the Fourier transform, is applicable only to linear systems; however, the linearity of the Magnavolt appears to be good enough for a transform to yield worth-while results. The method employed is described by Tustin[1] and Henderson.[10] The machine described by the latter was available to carry out the computation, the method being very laborious when carried out arithmetically; other methods are preferred when no machine is available.[12]

The points on the locus diagrams in Figs. 3 and 5 obtained from the transformation will be seen to be very inaccurate, diverging considerably from the measured loci. The main reason for this inaccuracy is that the response of the Magnavolt is less linear than it appears. Inaccuracy of recording and measuring the step-function response, owing to the small scale of the records, are contributory factors.

A check was made on the accuracy of the method by applying the transformation to a simple exponential curve, the result being perfectly satisfactory.

(7) CONCLUSIONS

The results of the investigation show that, provided suitable methods are chosen to allow for the effect of feedback on the effective resistance of the machine, the measured values of resistance, inductance and voltage constants may be used to predict the frequency and step-function responses within the linear range. For the latter the accuracy can be improved by allowing for the phase shift introduced by hysteresis. The direct transformation from step-function to frequency response appears to be unsatisfactory in this case.

Good approximations to the time-constants are obtained from log-linear graphs of the step-function responses, and will enable the frequency response to be obtained indirectly.

An attempt has been made to test the machine as a 3-stage exciter, provision being made for the first stage to be used as an amplidyne. The results of static tests were fairly satisfactory, but dynamic tests were not possible, owing to the extremely non-linear nature of the response and the very large effects of hysteresis.

(8) ACKNOWLEDGMENTS

The author would like to record appreciation of the gift of the Magnavolt to Wolverhampton and Staffordshire Technical College by the English Electric Co., Ltd., and for information and assistance on technical points from members of the staff of that company. Thanks are also due to Mr. L. L. Ruderman, formerly Head of the Electrical Engineering Department at Wolverhampton, for permitting the use of the equipment, and to fellow members of the staff of the College for their advice, and finally to Professor Tustin for permission to use the harmonic analyser at Birmingham University.

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(10) APPENDICES: EFFECTS OF UNEQUAL AIR-GAP RELUCTANCES

(10.1) Feedback from Second-Stage Field

If positive m.m.f. produces a north pole, reference to Fig. 1 shows that

$$\Phi_{12} = \frac{2(F_2 + F_4)}{S_1 + S_2} \quad . \quad . \quad . \quad . \quad . \quad (4)$$

$$\Phi_{34} = \frac{-2(F_2 - F_4)}{S_3 + S_4} \quad . \quad . \quad . \quad . \quad . \quad (5)$$

Therefore the 2-pole field flux is

$$\Phi_2 = \Phi_{12} - \Phi_{34}$$

$$= 2\left[F_2\left(\frac{1}{S_1 + S_2} + \frac{1}{S_3 + S_4}\right) + F_4\left(\frac{1}{S_1 + S_2} - \frac{1}{S_3 + S_4}\right)\right] \quad . \quad . \quad . \quad . \quad (6)$$

The second term is a function of F_4 and represents a feedback from the field winding of the second stage, which can be positive or negative depending on the relative values of the gap reluctances.

(10.2) Feedback from First-Stage Armature

The total armature m.m.f. for the first stage acts across S_1 and S_2 in parallel, in series with S_3 and S_4 in parallel.

$$\text{Thus,} \qquad \text{total flux} = \frac{F}{\dfrac{S_1 S_2}{S_1 + S_2} + \dfrac{S_3 S_4}{S_3 + S_4}} = \Phi_A \quad . \quad . \quad (7)$$

The fraction of this flux passing through pole 1 is

$$\Phi_A \frac{S_2}{S_1 + S_2} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (8)$$

and that passing through pole 4 is

$$\Phi_A \frac{S_3}{S_3 + S_4} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (9)$$

Therefore the component of armature flux in line with the input field is

$$\Phi_A\left(\frac{S_2}{S_1 + S_2} - \frac{S_3}{S_3 + S_4}\right) \quad . \quad . \quad . \quad . \quad (10)$$

Which, on substitution for Φ_A, becomes

$$F\left[\frac{S_4 S_2 - S_1 S_3}{S_1 S_2 (S_3 + S_4) + S_3 S_4 (S_1 + S_2)}\right] \quad . \quad . \quad (11)$$

This shows that inequality of air-gaps leads to a feedback component of armature flux which can be positive or negative according to the relationship of the gap reluctances.

DISCUSSION ON
'ELECTROLYTIC PROCESSES FOR SURFACE CONDITIONING OF METALS'*

EAST MIDLAND CENTRE, AT DERBY, 5TH MARCH, 1957

Mr. L. E. Probert: I agree that electrolytic means are indispensable for cleaning pretreatments prior to electrodeposition, and also for anodic etching for material inspection. I endorse the author's remarks relating to the dangers of hydrogen embrittlement with certain high tensile steels. Unfortunately, we are not always able to manage without the scrubbing action of hydrogen evolution at the cathode. I was interested to hear that cathodic anodic reverse polarity as opposed to cathodic treatment alone tends to reduce the danger of absorption of hydrogen, but in my experience it does not do enough to allow full use on all steels.

In my organization we find that electrolytic descaling is not required. Instead, we use hydrochloric acid or dry blasting on ferrous materials, and hydrofluoric ferric sulphate on stainless steels and some nickel alloys.

Referring to anodizing, I was interested in the reference to chromic acid being immune from retention in inaccessible areas or in material pores. We find that chromic acid anodizing is the best standard form, and that a sealing treatment is not necessary, the anodic film itself being very corrosion resistant; we only use a dichromate seal for prevention of intercrystalline penetration during heat treatment. There is, of course, the very important advantage of being able to inspect metallurgically after anodizing in chromic acid, chrome seepage indicating areas of trouble. On our aluminium-alloy forgings, chromic anodizing is the primary inspection aid.

On the question of dangers from retained electrolyte, I remember evidence of this in the case of sulphuric anodizing which caused us some trouble several years ago, in spite of precautions such as thorough swilling and oven drying.

There is no doubt a need for more erosion-resistant films than the standard coatings, and in consideration of the film produced by Dr. Campbell, which is thick and hard, we have effected a compromise by producing a film, not as hard or as thick, but it does not employ such low temperatures, nor is the effect on fatigue strength so marked. Fatigue drop cannot always be ignored, and all thick anodic films suffer somewhat in this respect.

With regard to electropolishing, we have a very high opinion of electrolytic polishing, but a lot of work is needed on the

* CUTHBERTSON, J. W.: Paper No. 1866 U, October, 1955 (see **102** A, p. 501).

various components prior to putting the job on a production basis. Provided this can be done, the time saved, with the production of an inspectable surface, is immense. An example of the work involved is turbine blade polishing. By the old method, involving hand polishing to a mirror finish so that anodic sulphuric etch for material inspection could be carried out, a considerable amount of time was spent on a finish which was required, not for engine performance, but for preparation of the surface for inspection. After a lot of work, largely by trial and error, we produced a set-up which enabled us to electropolish a large number of blades at a time, producing at the same time a surface for inspection. This involved conforming cathodes, with correct current, temperature and solution conditions.

In the decorative field, there is doubt about electrolytic polishing, owing to the necessity for having specially clean material from the suppliers before the very good lustre can be employed, since the smallest defects are revealed.

Mr. W. A. Sutton: The author said that cathodic cleaning should be avoided since this is conducive to embrittlement, but there are cases where a combination of anodic and cathodic cleaning is essential. Surface conditions occasionally occur which on electroplating give unsatisfactory adhesion without pretreating cathodically. This effect precludes an electrodeposited coating on high tensile materials where hydrogen embrittlement is dangerous.

I understand the author to say that a combination of anodic with cathodic treatment in the precleaning minimized the danger of embrittlement. Are there any grounds to substantiate this? On solution composition we have found additions of cyanide indispensable in facilitating the solution of oxides.

On anodizing an interesting point made was in connection with the active layer formed on the surface—hydrated aluminium oxide—which constitutes the basis of film build-up. Does this mean that this layer is conductive without porosity?

Mr. Probert dealt with potassium-dichromate sealing. This sealing effect seems to be due in part to deposition of chromic oxide in the film. Dichromate-sealed anodized high-temperature parts from engines show the coloration of chromic oxide.

In connection with sulphuric anodizing we experienced trouble on paint lifting resulting from delayed sulphuric-acid oxidation from casting pores.

The author mentioned the use of alternating current. I would ask how indispensable he thinks alternating current is in the formation of hard thick films. Work we did on oxalic anodizing with direct current produced films on one alloy thirty times as erosion-resistant as the chromic film.

With regard to electropolishing and the viscous layer theory, we experienced a condition on aluminium which perhaps demonstrates the high concentration of dissolved salts in the hollows. This condition was such that deposition of salts occurred in the hollows formed between the blade form and the root.

Prof. J. W. Cuthbertson (*in reply*): In reply to Mr. Probert, it is appreciated that in some cases cathodic treatment may be essential. In such circumstances alternate anodic treatment may reduce the danger of hydrogen embrittlement, but it is agreed that some danger may still persist with certain steels. Subsequent heat treatment to remove the hydrogen is not always practicable, and can sometimes have an adverse effect on the plating.

Where thorough rinsing is not possible, the chromic-acid anodizing process should always be considered in preference to the sulphuric-acid process. The advantages of chromic-acid anodizing from the inspection viewpoint are very real; the fact that the coating it produces need not be sealed is also in its favour. The modified hard anodizing process to which Mr. Probert refers appears to be attractive, as the effect of anodizing on fatigue strength cannot always be ignored.

I believe that electrolytic polishing has a considerable future. For decorative finishing it is certainly necessary for the basis metal to be of a good quality, and this is now being increasingly recognized. In this, as in all applications of the process, success depends on its proper application, and control of the variables to which Mr. Probert refers is most important.

Mr. Sutton's point on hydrogen embrittlement has been partly dealt with above, and I can only add that practical experience indicates that the combined anodic-cathodic treatment can be less dangerous than pure cathodic treatment. Concerning anodizing, the so-called active layer is close to the basis metal, and may be regarded as an embryonic layer from which the alumina film proper grows. The evidence is that the thickness of this layer does not change with time, and that it is not a barrier to the passage of electricity. I am of the opinion that possibly this layer is non-porous, so that conduction through it may be electronic rather than electrolytic. Superimposed alternating current reduces stress in many electroplating processes, but I cannot say whether there is any parallel effect in anodizing. The properties of anodic films vary with the method by which they are produced, and I am interested to note the marked increase in erosion resistance that Mr. Sutton has obtained by direct-current treatment in oxalic acid. I have always been surprised, in view of my own knowledge of the oxalic acid process, that it has not been more widely adopted.

Mr. Sutton's comments on the theory of electrolytic polishing are noteworthy. The viscous layer theory offers a reasonable explanation of the mechanism of electrolytic polishing; diffusion plays an important part in controlling the smoothing action.

DISCUSSION ON 'FIRES OF ELECTRICAL ORIGIN'*

Mr. N. S. Bryant (*communicated*): Cables having polythene insulation and p.v.c. sheath to B.S. 1557 are now being used extensively in house wiring. As polythene is inflammable and liable to softening on overheating should not tests on this type of cable be carried out? Furthermore, vulcanized-rubber-insulated lead-alloy-sheathed cables have been used extensively for house wiring and many such installations are still in use. For tests to be complete should not this type also be included?

With reference to Section 8.1.1, should not the test conditions approximate as closely as possible to normal installation conditions? These would be:

1. *Single core, vulcanized rubber insulated, taped, braided and compounded cables and p.v.c. insulated (unsheathed) cables.*

Two or more cables in conduit, the conduit size being in accordance with Table 11 of The Institution's Wiring Regulations, 13th Edition.

* LAWSON, D. I., and FRY, J. F.: Paper No. 2316 U, April, 1957 (see 104 A, p. 185).

2. *Twin flat, insulated and sheathed cables. All types.*

Fastened to softwood joists by buckle clips.

What voltage is maintained between the cores of twin cables under test? In practice this is 200–250 volts. When a short-circuit occurs between the cores, local heating at the point of contact may cause a fire to break out. The tests would therefore be realistic if mains voltage were maintained between the conductors.

Should not the current ratings given in the reporting form (Section 8.2) conform with those given in Tables 12 and 13 of the Wiring Regulations? As immersion heaters rated at 3000 watts are in common use should not the rating given be extended to include this value?

Mr. R. H. Dean (*communicated*): The authors are to be congratulated on a paper which gives a very great deal of information not previously tabulated, and their forecast of the future trends of the incidence of fires is significant and admonitory. However, the incidence of these electrical fires depends on the method of protection of cables and apparatus.

On this point a fundamental difference of approach exists on the North American continent. There, the authorities insist that the fuse links, which are of the cartridge or plug type, shall operate on an overload of 35% within specified times depending on the rating. In Britain, wiring regulations make no specific requirements. An analysis of the North American requirements shows that they correspond very closely to our B.S. 88: 1952 Class P. Reference is made in the paper (Appendix 8.1.2) to the size of fuse customary in domestic premises. This is quoted at 60amp, and the authors readily point out how soon fires can be started in cables at this current. This does not, however, present the whole of the picture. If the 60amp fuse is the semi-enclosed (copper wire) type it would be expected to carry 120amp almost continuously, and if of the silver-wire cartridge-fuse link type (B.S. 88 Class Q), up to 105amp continuously. A Class P with a minimum fusing factor of 1·25, i.e. 75amp, approximates in this respect to North American requirements. Have the authors access to any statistics from North America which might show the influence of fusing factor on the incidence of fires, and have they any comments to make on this difference of outlook between Britain and North America?

Messrs. D. I. Lawson and **J. F. Fry** (*in reply*): Mr. Bryant suggests that the mains supply voltage should have been maintained between the cables under test, to investigate any fires due to short-circuits. This was done, though it was not made clear in the paper. The experimental arrangement is shown in Fig. A. The two low-voltage windings of the transformer were connected to the cable under test and the conductors of the cable were connected across the mains supply through a 15amp fuse. It does not, however, appear from our experiments that fires start as a result of current flowing between the conductors of cables, and the fuse only operates when the conductors actually touch, forming a short-circuit, in which case the power dissipation is small.

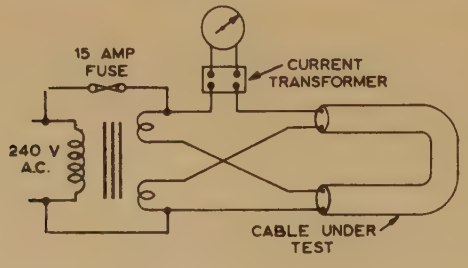


Fig. A.—Wiring diagram for cable experiments.

A limited series of experiments have been carried out to obtain information on the p.v.c.-sheathed polythene-insulated cables and on lead-covered rubber-insulated cables mentioned by Mr. Bryant; these are summarized in Fig. B. It will be seen that the curves are similar to those already quoted. We do not think that cables enclosed in conduit would give rise to fire.

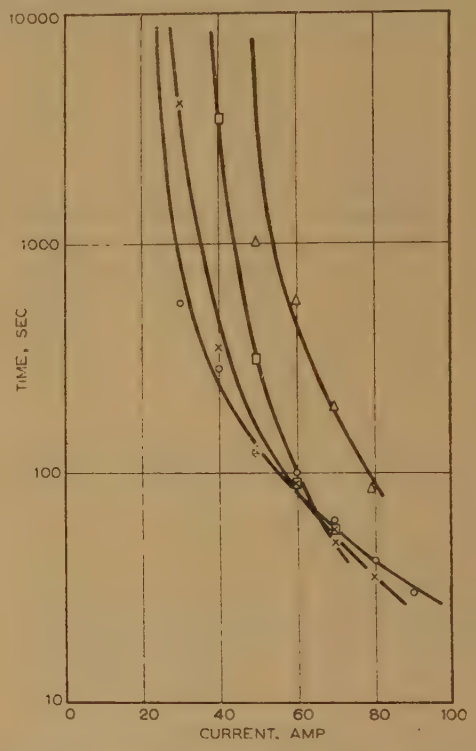


Fig. B.—Graph showing the time for either a fire or a short-circuit to develop on a cable when carrying various currents (cf. Fig. 4 on page 188).

○—○—○ Polythene-insulated p.v.c.-sheathed 3/·029 in cable. Short-circuit.
△—△—△ Polythene-insulated p.v.c.-sheathed 3/·029 in cable. Fire.
×—×—× Rubber-insulated lead-covered 1/·044 in cable. Short-circuit.
□—□—□ Rubber-insulated lead-covered 1/·044 in cable. Fire.

We agree that the circuit ratings given on the reporting form would be better if they were amended to conform with The Institution's Wiring Regulations.

The ratings for immersion heaters are intended as a guide and the more common rating is 2000 watts. We will, however, draw the attention of reporting staff to the fact that the 3000-watt type is sometimes encountered.

Mr. Dean mentions the possible influence of the fusing factors current in North America and in this country on fires of electrical origin. This is an interesting point and one of which we shall endeavour to take account in the current survey, though at present we have no statistical information on electrical fires in North America.

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The Institution of Electrical Engineers
Paper No. 2413
Dec. 1957
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ARC MOVEMENT DUE TO THE MAGNETIC FIELD OF CURRENT FLOWING IN THE ELECTRODES

By A. E. GUILE, Ph.D., B.Sc.(Eng.), Associate Member, and S. F. MEHTA, B.E.

(*The paper was first received* 13*th April, and in revised form* 12*th July*, 1957.)

SUMMARY

The fact that an arc moves in the magnetic field of the current flowing in the electrodes towards the arc roots is often used to control its position or to help to bring about its extinction.

The paper describes tests made to investigate the factors which affect the movement of an arc between parallel electrodes, fed with current at one end, in air at atmospheric pressure. Some of this movement is quite random and could never be predicted, but much of it—particularly that of the cathode spot—is consistent and regular. The way in which this regular movement of d.c. and a.c. arcs varies with cathode material and electrode spacing has been investigated at currents up to 3 000 amp a.c. and 500 amp d.c.

The results show that arc movement cannot be calculated by considering the arc as a physical conductor acted upon by electromagnetic driving forces and aerodynamic retarding forces. The movement is primarily a process of transfer of electron emission along the cathode surface, and is therefore concerned with electrode surface phenomena as well as with movement of the arc in the air between the electrodes.

(1) INTRODUCTION

Recent papers on the movement of arcs in magnetic fields[1, 2] stress that, although there is wide application of magnetic fields to arc control, there is very little information available either on quantitative results or on discussions of the mechanisms involved. Much of the work which has been done was concerned with the retrograde movement of an arc in a magnetic field at low pressures, and most of that done at atmospheric pressures has dealt with arc motion in an externally applied transverse magnetic field. Winsor and Lee[1] made tests on a variety of non-ferrous electrodes in magnetic fields up to 0·02 Wb/m^2 and at direct currents up to 100 amp; they took cathode-spot velocity as the criterion for arc velocity, owing to discontinuous motion of the anode spot, and gave velocities measured photographically for various currents and magnetic fields at an electrode spacing of $\frac{1}{8}$ in. Rieder and Eidinger[2] also measured arc velocity in similar ranges of current and magnetic field on non-ferrous electrodes at spacings from $\frac{1}{8}$ to $1\frac{1}{4}$ in. These investigators[1, 2] found that regular arc velocity appeared to be independent of cathode material. Babakov[3] used higher currents and magnetic fields (up to 400 amp and 0·1 Wb/m^2) but very small spacings (up to $\frac{1}{8}$ in): he found some effect of electrode material on arc velocity and concluded that displacement of the cathode spot plays a vital role.

In all these tests, in addition to the externally applied transverse magnetic field, there was a further component of magnetic field inside and outside the electrodes, owing to the current flowing in them towards the arc roots. This has frequently been neglected[1, 2] but was included by Babakov.[3] For arc currents of the order of 100 amp the increase in magnetic field near the surface of the electrodes is not negligible. Little attention has been paid to consideration of arc movement due solely to the presence of the field of the current in the electrodes, and when it has been considered,[4, 5] very few quantitative results have been given. Hochrainer[4] set up heavy-current a.c. arcs between electrodes sloping downwards; the current feed was at the top, so that the magnetic forces tended to make the arc move downwards while the thermal forces had the opposite effect. Freiberger[5] measured some velocities of heavy-current a.c. arcs between busbars.

In the experiments now reported the only magnetic field was that produced by current flowing in the electrodes, and ferrous electrodes, hitherto neglected, were investigated. The results show that the motion of an arc in a magnetic field depends upon such factors as cathode material and surface, and that it moves faster on ferrous electrodes than on non-ferrous ones. This last result, which depends on the magnetic properties of the cathode, emphasizes the dependence of arc movement on processes occurring at the cathode surface.

Tests are described on a.c. arcs up to 3 000 amp and d.c. arcs up to 500 amp, with the electrode spacing varied between $\frac{1}{2}$ and $7\frac{1}{2}$ in. The electrodes used were of mild steel, galvanized steel, brass, copper, aluminium, Duralumin, zinc and carbon, and were rods of $\frac{3}{8}$ in diameter connected at one end to the supply circuit

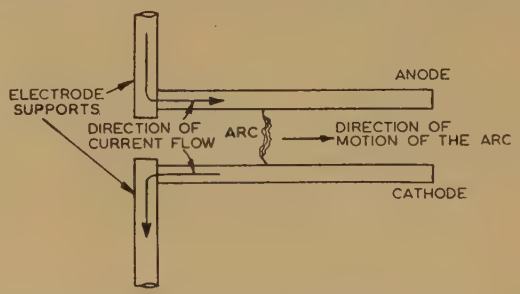


Fig. 1.—Electrode arrangement.

(Fig. 1). The arc was initiated by 6 amp fuse wire about 1 in from the feeding end of the electrodes, and was photographed at 960 frames/sec using a rotating-drum camera.

(2) MOVEMENT OF AN A.C. ARC UNDER ELECTROMAGNETIC FORCES IN STILL AIR

All a.c. tests were made on the same pair of parallel mild-steel electrodes which were not cleaned in any way, so that their surfaces were well pitted; the electrodes were 2 ft long and were mounted horizontally, one above the other. The closing of the circuit was controlled so as to minimize the direct component of the current and the frequency was 55–50 c/s.

Photographs showed that if the arc column happened to touch one of the electrodes, the arc quickly transferred itself to a new root formed at this point of contact, since this offered a path of lower impedance and the negative slope of the voltage/current characteristic caused instability. This unpredictable behaviour is here termed 'random movement'. Throughout the paper, all arc movements considered and all velocities measured (except in the presence of a wind) are those which occurred in regions where there was no random movement.

Written contributions on papers published without being read at meetings are invited for consideration with a view to publication.
Dr. Guile and Mr. Mehta are in the Electrical Engineering Department, Queen Mary College, University of London.

It was observed that the cathode spot ran ahead of the anode spot, so that the arc moved with one root leading, first on one rod and then on the other. In order to measure a consistent quantity, therefore, an arc velocity was determined between two frames in each of which the two roots were almost exactly opposite each other. The distances over which the velocity was measured were as long as possible, and were usually several inches, the limit being set by the necessity for finding such lengths of travel where there were no random jumps. The arc velocity is plotted against arc current in Fig. 2 for an inter-electrode spacing of $1\frac{1}{4}$ in;

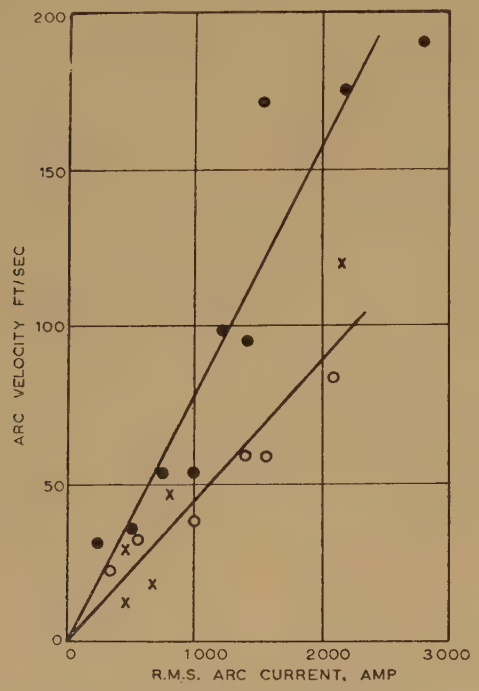


Fig. 2.—Velocities of a.c. arcs on mild steel, copper and brass electrodes, $1\frac{1}{4}$ in apart in still air.

● Mild steel.
○ Copper.
× Brass.

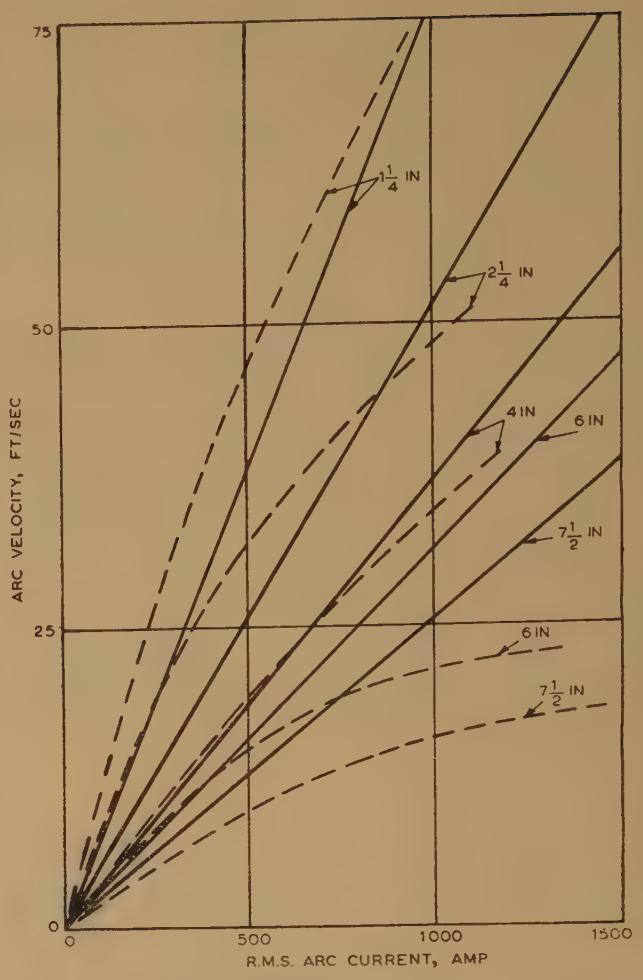


Fig. 3.—Comparison between velocities of a.c. arcs in still air and equilibrium wind speeds, for steel electrodes at various spacings.

———— Arc velocity in still air.
— — — — Equilibrium wind speed.

a straight line may be drawn as a reasonable approximation to the best curve through the points, and this is true for the other spacings tested, namely $2\frac{1}{4}$, 4, 6 and $7\frac{1}{2}$ in (Fig. 3). The slopes of these lines plotted against electrode spacing yield a curve, the equation to which is

$$\frac{v}{I} = 0{\cdot}016\left(\frac{5}{d}+1\right) \quad . \quad . \quad . \quad . \quad . \quad (1)$$

where v = Arc velocity, ft/sec.
I = R.M.S. arc current, amp.
d = Electrode spacing, in.

By means of eqn. (1) the arc velocity for any given current and mild-steel electrode spacing in the range tested may be calculated.

(3) MOVEMENT OF AN A.C. ARC WITH WIND OPPOSING THE ELECTROMAGNETIC FORCES

The electrodes were placed in a wind tunnel with a 16 in square insulated working section so that the wind was directly opposed to the magnetic forces. For a given current and rod spacing, a wind speed was determined at which the arc tended to remain stationary; this 'equilibrium wind speed' is plotted in Fig. 3, which shows that it does not vary linearly with current but has a decreasing slope as current increases. The wind speed required to keep an arc with a given current stationary decreases with increase in the arc length, since the effective area of the arc is increased and it can more readily be blown back on to the rods to form new roots downstream.

When an arc was subjected to wind, that part of it near the root tended to move upstream under the magnetic forces, but the column was blown downstream by the wind, and by making contact with one of the electrodes, formed new roots downstream. Under wind conditions it is hardly possible to divide the arc movement into regions of regular travel and random movements, as can be done in still air, since the latter occur continually, particularly at the end of each half-cycle.

There are a number of differences in the character of the magnetic and wind forces. The magnetic field decreases rapidly with increased distance from the electrodes and its intensity varies widely during the cycle of current, while the wind velocity remains constant. The constituents of the arc column do not move forward as a body of gas under the magnetic forces, which act only upon the ionized particles, so that the motion of the arc does not necessarily represent the motion of the main body of the gas, but is rather a deflection of the point of maximum ionization.[6] Experiments by Dufour[7] showed an arc velocity of 2100 ft/sec while the mass of gas moved at 230 ft/sec. In

view of these differences in the behaviour and effects of magnetic and wind forces, it is hardly to be expected that the equilibrium wind speed would be the same as the arc velocity in still air, but Fig. 3 shows that they are substantially equal at shorter gaps and lower currents. Photographs show that the reason for the similarity with these gaps and currents was that when subjected to wind the cathode moved upwind at about the same rate as it does in still air. At the end of the half-cycle during which the cathode moved upwind a certain distance, the new anode formed about the same distance downwind from the last position of the cathode, because the column was blown back to the rod. At higher currents, and therefore higher equilibrium wind speeds, the cathode did not move so fast in a wind as in still air, perhaps because the wind was then able to influence the part of the arc extremely close to the electrodes. Less backward movement of the new anode position was therefore necessary to keep the arc in one place, so that this occurred at a wind speed less than that of the arc in still air.

The study of the movement of an a.c. arc is complicated by the following factors:

(*a*) The magnitude of the force on the arc varies cyclically.
(*b*) At a current zero the arc must restrike, so that anode and cathode spots transfer to the opposite electrode not necessarily reforming at the places which were active at the end of the previous half-cycle.

In order to eliminate these variable factors and to investigate the differences between anode and cathode movements, tests were made with direct current. It is not considered that any exact correlation can be made between the cathode velocity with direct current and the arc velocity with alternating current. As is shown in the next Section, the cathode velocity for a given current does not have a single value on pitted electrodes and the anode movement is very variable. Furthermore, factors (*a*) and (*b*) make comparison difficult. The former effect (*a*) is illustrated in Fig. 4, where the velocity of the root on one electrode is plotted

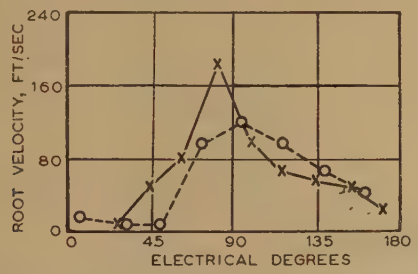


Fig. 4.—Variation of root velocity on one electrode during two successive half-cycles of current.
———— Cathode.
— — — — Anode.

for two successive half-cycles, i.e. one curve represents cathode velocity and the other anode velocity. The curve for the cathode is typical of many which were obtained, but the anode was rarely found to show such consistent behaviour.

(4) CATHODE MOVEMENT WITH D.C. ARCS

(4.1) Pitted Mild-Steel Electrodes

The first tests with direct current were on the same horizontal mild-steel electrodes which had been used for some 150 a.c. tests, and these rods were not cleaned at any time. The tests showed that for a given current the cathode velocity varied widely, as shown in Fig. 5. This result was surprising, in view of the fact that the velocity of the whole a.c. arc varied consistently with current, as shown in Fig. 2, and this emphasizes the necessity for

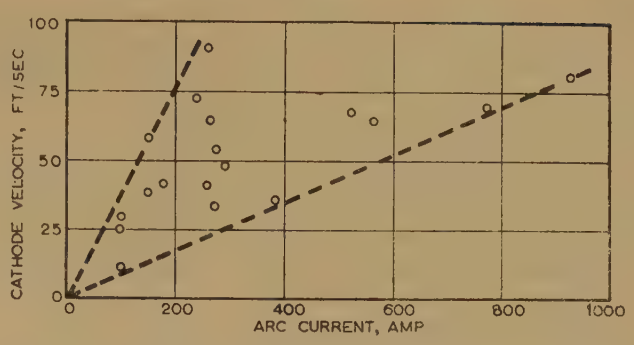


Fig. 5.—Cathode velocity on pitted mild-steel electrodes, 1¼ in apart.

measuring such a velocity over the longest possible times and distances, since it is concerned with many variable factors. It was considered that this wide variation of cathode velocity was due to varying surface conditions along the rod. For this reason, in all subsequent tests described in the paper (except in the case of zinc, where the electrodes were turned from rectangular-section bar), each cathode was a drawn rod of ⅜ in diameter in a clean but untreated condition, and only one test was made on each cathode.

(4.2) Clean Mild-Steel Electrodes

(4.2.1) Horizontal Electrodes.

On clean rods the cathode velocity was found to be lower than on pitted rods, and the movement was regular and consistent. The results were in marked contrast, e.g. on pitted rods the cathode velocity of a 100-amp arc varied between 10 and 30 ft/sec, whereas on a clean cathode a 100 amp arc travelled in a perfectly regular straight track, 12½ in long, with a velocity varying only between 2·4 and 2·9 ft/sec. When a rod was cleaned with very fine emery paper and well polished by buffing, the cathode track at 100 amp tended to follow a helix in the direction of buffing, and although the velocity was constant over much of the track at 3·5 ft/sec, it did reach 15 ft/sec in places. This illustrates the dependence of cathode velocity on surface conditions. It was found, however, that, if no treatment was given

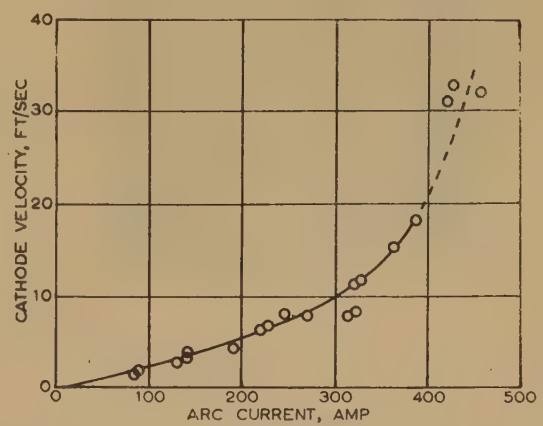


Fig. 6.—Cathode velocity on clean mild-steel electrodes, 1¼ in apart.
———— Regular track.
— — — — Semi-regular track.

to new drawn rods, consistent velocity variation with current was found, as shown in Fig. 6.

Before proceeding to discuss cathode velocity further, it is necessary to describe certain salient features of the cathode tracks; four types of tracks were found, namely

(*a*) A track which was continuous* and uniform in width and appearance, usually deep blue in the centre and dark brown at the edges. These tracks were found to extend as much as 12 in along the cathode, but most of them were between 0·5 and 2 in long. Such a track of about 0·5 in or more is referred to as a 'regular track', and the type of movement as 'regular movement'.

(*b*) The entire cathode travel was composed of small and large jumps from point to point, and very small cathode spots of different sizes were scattered at random along the rod. These patches showed great variation of colour from deep blue to pale straw. Motion in this region is referred to as 'random movement'.

(*c*) A track intermediate between (*a*) and (*b*), being partly regular and partly random. There were sections of uniform track which did not exceed $\frac{1}{2}$ in in length, and were generally between $\frac{1}{8}$ and $\frac{1}{4}$ in, more straw than blue in colour. These small sections were lined up to form a long track with small gaps between them where random jumps occurred. This type of movement is not so well defined as type (*a*) as it is a combination of (*a*) and (*b*) which occurs near the onset of random movement. This is referred to as a 'semi-regular movement or track'.

(*d*) A continuous track, light grey in colour, with a tree or branch appearance. Microscopic examination showed that the electrode surface was not damaged appreciably, merely discoloured. Occasionally a very small area where melting occurred could be seen, but there were sections of track where no sign of melting could be detected with magnification of 1 000. This type of track has been found to occur only at cathode velocities greater than those corresponding to tracks (*a*) and (*c*), so that it is referred to here as 'high-speed regular track'.

Fig. 7 shows, on five successive frames covering a total time of 5·2 millisec, a 230 amp arc between clean steel electrodes $1\frac{1}{4}$ in apart. The photograph shows the cathode root advancing

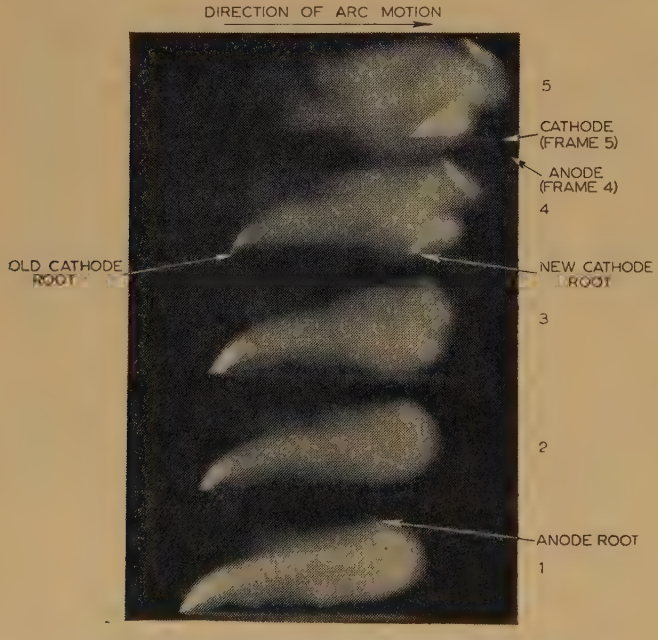


Fig. 7.—D.C. arc moving between parallel steel electrodes.

gradually along the electrode; the arc column then touched the cathode, forming another root some way ahead, and subsequently the whole arc current transferred to this new root. Very uniform and well defined cathode tracks were found before and after this random jump of $2\frac{1}{4}$ in. Owing to random jumps, regular cathode tracks cannot make up the whole of the distance travelled by the arc. Table 1 shows that the percentage of the aggregate of regular tracks to the gross distance travelled by the arc tends to rise with increase in spacing between the electrodes, from about 15% at $\frac{1}{2}$ in spacing to 60 or 70% at spacings of 3 and $7\frac{1}{2}$ in. Since the term 'regular track' has been used only for tracks of $\frac{1}{2}$ in or more in length, the aggregate of all regular tracks will be less than that of all net cathode tracks. Table 1 shows that the cathode track is often regular for as much as 90% of its total or net track length.

When the current, and therefore the cathode velocity, is increased, the cathode movement changes first from (*a*) to (*c*) and then to (*b*). In some cases type (*d*) appears between types (*b*) and (*c*).

For a $1\frac{1}{4}$ in electrode spacing the cathode movement is regular for currents up to about 350 amp, and between 350 and 400 amp the boundary region between regular and semi-regular movement was found, semi-regular movement extending up to currents of about 470 amp; at greater currents all movement was random. Fig. 6 shows the high consistency of cathode velocity measured over regular tracks. This consistency is emphasized by the scatter in cathode velocity shown in Fig. 5 and by the wide variation in results for various materials and conditions shown in Table 2. Fig. 6 shows that points in the current range 420–470 amp, which relate to semi-regular movement, appear to lie on a continuation of the curve for regular velocities. The velocities for semi-regular tracks were obtained by measuring considerable distances along such tracks and deducting therefrom the aggregate of all the jumps.

The family of curves for mild-steel rods (Fig. 8) at various

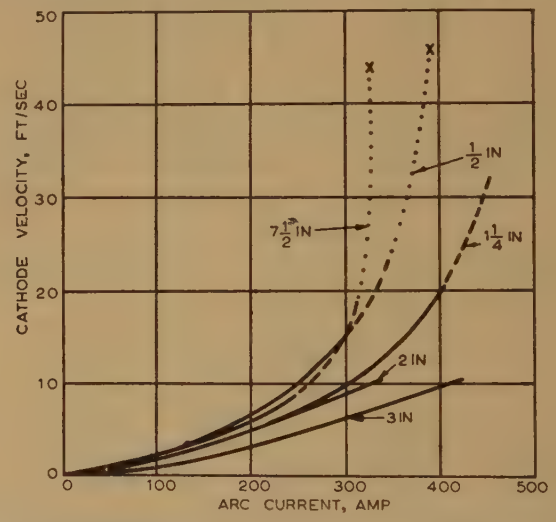


Fig. 8.—Cathode velocity on mild-steel electrodes at various spacings.

———— Regular track.
— — — — Semi-regular track.
× High-speed track.

spacings shows that there can be no very simple explanation of the way in which regular cathode velocity varies with electrode spacing. There are a number of factors influencing this, some of which must be conflicting, since there is a general tendency for the velocity at a given current to fall as the spacing is increased from $\frac{1}{2}$ to 3 in, but at $7\frac{1}{2}$ in the velocity rises again. Although, as stated earlier, the arc motion is not that of a definite mass of gas, so that resistance to motion in the aerodynamic sense does not apply, it does seem reasonable to expect velocity to fall with increase in spacing, so that the increase in velocity at a $7\frac{1}{2}$ in spacing is somewhat anomalous.

It was stated earlier in this Section that, as the electrode spacing is increased, there is a tendency for more of the cathode movement to be regular. Part of the explanation for this may be that the boundary between two types of cathode movement,

* Microscopic examination shows that what appears to be a continuous track is made up of separate emitting areas, although evidence of this may be somewhat obscured if considerable melting has taken place. Some tracks which appeared continuous have needed magnification by electron microscope before the discontinuities can be seen.[8] The tracks strongly suggest that there are multiple emitting points in each cathode spot.

Table 1

CATHODE TRACKS

Spacing between electrodes	Distance travelled by cathode*				Maximum current	Regular movement / Gross movement		Regular movement / Net movement	Gross average velocity
	Net	Gross	Regular	High-speed regular		Measured	Average		
in	in	in	in	in	amp	%	%	%	ft/sec
½	7·7	19·2	7·0		110	36	17	91	6
	10·3	22·0	4·0		360	18		39	29
	17·0	21·5	2·5		400	12		15	26
	11·0	22·0	0·8		690	3·5		7	40
1¼	6·2	19·75	6·0		150	30	38	96	9·0
	9·7	22·0	7·3		250	33		75	14·5
	11·0	22·0	8·7		320	39		79	14·5
	12·6	22·0	11·8		410	54		94	16·5
	8·5	22·0	7·6		500	34		90	21·5
2	10·0	14·7	10·0		115	68	55	100	3
	14·6	22·5	14·6		125	65		100	3·8
	16·3	22·0	13·0		250	60		80	8
	10·0	23·0	6·0		320	26		60	15
3	8·2	10·4	8·2		150	79	70	100	2·2
	20·0	23·0	18·5		165	81		92	11·5
	14·2	22·5	13·8		230	61		98	6·6
	18·4	23·5	13·6		230	58		74	8·5
7½	10·1	10·1	10·1		130	100	63	100	2
	3·5	3·6	3·5		155	97		100	1·5
	13·2	15·6	3·4		270	22		26	10
	14·4	18·3	6·0	7·6	325	63		93	11
	3·1	5·2	1·6		340	31		52	2·9

* Net cathode distance is aggregate of all tracks.
Gross cathode distance is the total distance travelled by the cathode, including jumps.
Regular cathode distance is the aggregate of all regular tracks, i.e. those of 0·5 in or longer.

such as that between regular and semi-regular movement, is determined more by a limiting velocity than by a limiting current. Support for this is given by the fact that, whereas high-speed regular movement has been observed only at velocities between 45 and 55 ft/sec, the current at which it occurred was in several cases nearly 400 amp and in another case only 115 amp. Fig. 8 shows points of high-speed regular movement on ½ and 7½ in gaps at currents between 300 and 400 amp. A slot, $\frac{3}{32}$ in × $\frac{3}{32}$ in, was machined along the axis of a mild-steel electrode. A ⅛ in thick ⅜ in bore Paxolin sleeve with a slit ⅛ in wide along its axis, was placed over the electrode with the opening in the sleeve over the slot. This electrode was made the cathode and was placed 1¼ in from the anode with fuse wire joining the latter to the base of the slot. A high-speed regular track was found along the base of the slot with no burning on the sides of the slot (see Table 2). The velocity here was close to that in the other similar tracks, but owing to an increase in magnetic field strength near the root, caused by the slot, was obtained at a current of only 115 amp.

This increase in regular movement with increase in spacing also depends upon the position of the arc column relative to the electrodes, since it is when the column touches an electrode that a random jump occurs. The part of the arc within about 1 cm of the electrodes (sometimes called the "root' portion), is narrow and well defined and appears to have a certain 'stiffness' or 'elastic' nature. This root part of the arc does not appear to change much with change in electrode spacing. When the root on one electrode leads the other, the arc is curved as though there were a minimum radius of curvature where the root merges into the comparatively diffuse column. Owing to this curvature, which to judge from the films, does not appear to change much with spacing, the arc column can more easily touch the opposite

Table 2

TESTS MADE ON PARALLEL RODS 1¼ IN APART

Material	Condition	Current	Cathode velocity		
			Average	Minimum	Maximum
		amp	ft/sec	ft/sec	ft/sec
Copper	Untreated	115	0·71	0·70	0·72
Aluminium ..	Untreated	125	0·80	0·76	0·86
Brass	Untreated	125	0·80	0·70	0·90
Galvanized steel	Untreated	125	0·35	0·21	0·48
Carbon	Untreated	160	0·10		
Mild steel ..	Untreated	100	2·5	2·4	2·9
Mild steel ..	After 150 a.c. tests	100	25	10	30
Mild steel ..	Buffed	100	6·1	3·5	12
Mild steel* ..	Buffed	100	19	11	25
Mild steel† ..	Machined	100	17	10	25
Mild steel‡ ..	Machined	115	54	50	64

All electrodes were of ⅜ in diameter, except for carbon, which was ½ in diameter.
* Second test four months after buffing.
† Travel on edge of slot.
‡ Arc at base of $\frac{3}{32}$ in × $\frac{3}{32}$ in slot; electrode surrounded by ⅛ in thick Paxolin sleeve with ⅛ in slot.

electrode and cause a random jump when the electrodes are close together. When Babakov[3] measured arc velocities on electrode spacings of 0·1–3 mm he found it necessary to repeat each test not less than six times and plot the average of the velocities obtained, whereas every point plotted on a graph in the present paper was obtained directly from a single test. This indicates that with the very short gaps tested by Babakov there was considerable scatter in the results. At greater spacings the

curved part of the arc column is far enough from the electrodes for it not to make contact without considerable lengthening first occurring.

Winsor and Lee[1] found regular cathode and anode tracks with copper electrodes, and when the current was raised the anode, or both anode and cathode, moved discontinuously. This is similar to results described in the paper, but there is a difference in that in their tests the average velocity in the region of random movement was only a fraction of that during regular movement, whereas reference to Table 1 and Fig. 8 shows that the gross average arc velocity is generally higher, owing to random movement, than if the arc had progressed regularly at the maximum current, e.g. at a 2 in spacing and maximum current of 320 amp. Fig. 8 shows that, even if the current had stayed constant at this value, the regular velocity could not have exceeded 9 ft/sec, whereas the gross velocity was 15 ft/sec.

Tests made with the top electrode as cathode or as anode showed no appreciable difference in velocity, but on the top electrode the root had a slight tendency to move upwards around the rod, owing to thermal or gravity effects. In most tests the lower electrode was therefore made the cathode.

(4.2.2) Vertical Electrodes.

Tests were made with the electrodes vertical at spacings of $\frac{1}{2}$, $1\frac{1}{4}$ and $7\frac{1}{2}$ in, and in each case it was found that the percentage of movement that was regular was similar to that with horizontal electrodes, and that the regular cathode velocities were the same as those at the same current with the arc moving horizontally. Hochrainer[4] reports that, on electrodes 43·3 in apart and inclined downwards at 60° with current feed at the top, a.c. arcs moved upwards at 1 400 amp and downwards for greater currents, i.e. that at this current magnetic forces overcame the thermal forces. His calculations on thermal and magnetic forces per unit arc length in the centre of the arc column do not, however, accord well with these observations, in that the former seem to be too small to overcome the latter. Even if an arc of 1 000 amp were assumed to have a radius as large as 5 cm and the density of the arc were assumed to be negligible, to give an upper limit to the thermal updraught, the upward thermal force per unit length at the arc centre would still be much less than the electrodynamic force.[4] The reason for this discrepancy between test results and force calculations is that macroscopic consideration of the arc as a physical conductor acted upon by electromagnetic forces in the air between the electrodes, and as a body of gas acted upon by thermal forces, does not represent the true conditions. High-speed photographs and microscopic examination of the regular cathode tracks make it clear that movement of the centre of the arc column does not affect regular movement, but can influence only the occurrence of random movement. The thermal forces on the arc near the electrodes appear to be negligible compared with the electromagnetic ones.[4] These considerations explain why regular cathode velocity is the same whether the arc is moving between horizontal or vertical electrodes, especially since it has been shown that the magnetic forces causing regular motion operate within the cathode spot itself.[9]

(4.3) Electrodes of Various Materials

Tests were made on $\frac{3}{8}$ in diameter rods of various materials. The zinc rods were turned to that diameter, but brass, copper, aluminium and Duralumin drawn rods with no further surface treatment were used. Fig. 9 shows that up to about 200 amp the cathode velocities on brass, aluminium and Duralumin were the same, while that on zinc was somewhat lower. These velocities on brass and aluminium were only about one-fifth of those for mild-steel electrodes. It has been shown that magnetic fields inside the electrodes can exert great influence on the

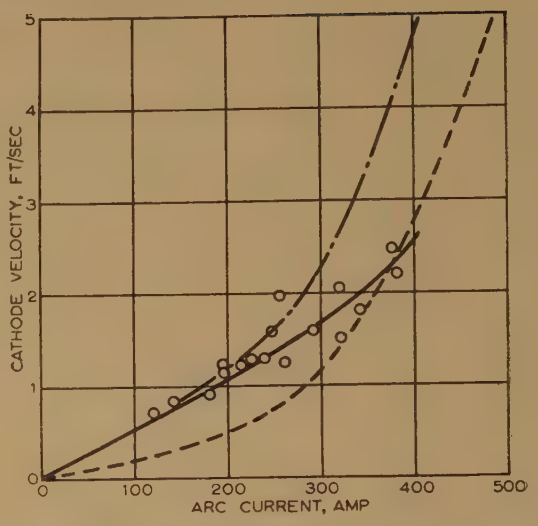


Fig. 9.—Cathode velocity on non-ferrous electrodes $1\frac{1}{4}$ in apart.

O O —— Brass.
— — — — Zinc.
—·——·— Aluminium and Duralumin.

movement of the arc,[9] so that the arc moves faster on steel than on non-magnetic cathodes, largely because the magnetic field within the electrode surface is greater in ferromagnetic electrodes. Above about 200 amp the cathode velocity on aluminium and Duralumin increases more rapidly than on brass, and regular tracks with consistent velocity variation are obtained up to 500 amp. On copper, the velocity is the same as on brass up to 200 amp, above which the nature of the track is very variable, although a track with very few random jumps is found. Parts of it appear somewhat similar to the semi-regular and high-speed regular tracks observed on steel. These changes in nature of the track on copper cause considerable variation in velocity for a given current, whereas for steel the velocity/current relation is consistent for types (*a*) and (*c*). The track on copper shows much less damage to the surface than for any other material tested, and even at 300 amp appears to be a discoloration with little evidence of melting. The cathode tracks on brass were extremely regular and gave very consistent results, as shown in Fig. 9. On aluminium and Duralumin the tracks were also extremely regular, the uniform straight track frequently extending the whole length of 7 in electrodes, This is in contrast to tests of Winsor and Lee[1] and of Stolt,[10] where pitting of the electrodes caused anchoring of the roots, so that results could not be obtained on aluminium. Very regular tracks with consistent velocities were obtained on zinc for currents up to 600 amp. Table 2 shows the cathode velocities for carbon and galvanized steel to be even lower than those on brass. On carbon electrodes the movement was irregular at currents above about 250 amp.

(5) ANODE MOVEMENT WITH D.C. ARC

The anode movement on steel electrodes was found to be much less regular than that of the cathode, since the anode had a greater tendency to remain anchored. Some regular anode tracks up to about 1 in long have been observed on steel, and a number of anode velocities were measured; these were found to be the same as, or sometimes slightly less than, the cathode velocities at the same currents. For the most part the anode travel was composed of small areas of different shape and size scattered at random along the rod. Occasionally the anode travelled by small regular jumps, forming very small circular patches at the places where it dwelt. These small patches were separated by uniform gaps and they lined up to form a long

track. The gross velocity in this region was high and the anode led the cathode by as much as 7 in. Several times this caused the anode to reform 6–7 in behind its previous position, owing to the column touching the rod. The tendency of the anode to jump ahead of the cathode on clean mild-steel rods was opposite to that on pitted rods, where, with 1¼ in spacing, the anode lagged behind the cathode by distances up to 10 in. The reason for this difference is that when the cathode velocity is comparatively low, as on clean electrodes, the magnetic field drives the arc column ahead of the cathode root, as may be seen in Fig. 7. Since the anode root makes most of its progress on steel by random jumps, its position depends largely on that of the arc column, and with the latter leading it is then able to jump ahead of the cathode root, whereas when the column lags behind the cathode root the anode root also lags.

On copper anodes there was some regular movement, but it was slower and less common than that on the cathode. Winsor and Lee[1] took cathode velocity as a criterion for arc velocity on copper and other materials, owing to intermittent motion of the anode. On aluminium the anode showed little tendency to move from one spot, but tracks up to about 1 in long have been observed. On brass the anode was almost as regular as the cathode and showed many regular tracks several inches long, the anode velocities falling on the same velocity/current curve as the cathode velocities. For currents between 300 and 500 amp the anode track became less regular, with sections about ¾ in long occurring at a few places. The tendency of the anode (top electrode) to move upwards around the rod was slight for most materials, but in a few tests with brass a number of sections of regular helical track were found, each section covering some 90–150°. The behaviour of the anode has great influence on the arc voltage: where cathode and anode both move regularly at the same velocity, as frequently happens on brass, the arc length remains fairly constant, so that the arc voltage does not vary much, e.g. on brass electrodes at 1¼ in spacing and 125 amp the arc voltage varied only between 65 and 95 volts. This contrasts with variation between 75 and 450 volts for mild-steel electrodes at the same spacing and current, which arises because the anode frequently anchors and then makes a random jump.

(6) DISCUSSION OF RESULTS

The experimental results described have shown that, when an arc moves because of magnetic fields set up by the current in the electrodes, the motion is determined chiefly by the movement of the cathode rather than that of the anode or column. Even when the arc is in a wind, it is the cathode movement which plays the chief part. The position of the anode and of the arc column has been found to cause only random jumps of the roots. The column often leads the roots and sometimes touches one or other electrode, thus forming a new root, but in many tests the cathode has a regular track extending along the whole length, so that the arc movement has been exactly the same as that of the cathode. The important process in the arc movement is therefore that of the transfer of cathode emission from point to point.

Further evidence for the small effect of the anode is that the cathode velocity on stainless steel is the same for both aluminium or stainless-steel anodes, although the anode spot anchors far more on the former than on the latter.[11] The fact that magnetic fields within the cathode spots have been shown to have great influence on the arc movement[9] makes it clear that, in any study of arc movements in a magnetic field, attention should primarily be focused on the emitting spot on the cathode. Even where an external transverse magnetic field is applied so that the field in the arc column is greater than in the experiments described here, there is evidence that the cathode movement is chiefly responsible for the arc travel.[1, 3]

The cathode emission processes for a stationary arc discharge are still a matter for controversy, so that it is not possible to advance detailed suggestions for the mechanism of transfer of emission due to a magnetic field. It would, however, seem that the magnetic field within the surface of the electrodes may cause galvanomagnetic and thermomagnetic effects in the micro-region of the solid-liquid-vapour transition which influence transfer of emission.[11] A process of this kind can account for the fact that distances between separate points of emission are so small that electron-micrographs are necessary to distinguish them,[8] and also for the fact that the cathode has been observed to move in an opposing wind with a velocity equal to or approaching its value in still air. For the larger spacings between emitting points which occur in some tracks, there may be a breakdown to the cathode surface from that part of the arc very near the cathode which is driven forward by the magnetic field. Freiberger[5] suggested that all arc movement in a magnetic field was due to this latter process, but the fact that cathode or anode spots move in a direction dictated by the magnetic field inside the electrode surface shows that such an explanation cannot account for regular cathode movement, although it does account for random jumps as suggested by Dunkerley and Schaefer.[12] Since, as has been shown here, the magnetic deflection of an arc is a very complex process, it is not surprising that different investigators have obtained results which may appear mutually inconsistent. There are many factors which influence the arc movement, e.g. electrode spacing, material, shape and surface conditions; current (and therefore associated magnetic field) and any externally applied magnetic field; and whether the velocity measured is that of the cathode or the anode, or is an average arc velocity. An example of the differences which are to be expected when so many factors are involved is that of change in arc velocity when the spacing between electrodes is varied. For gaps up to 1 mm Babakov[3] showed considerable change in velocity with electrode spacing, while there was less change between 1 and 3 mm, and Burghoff[13] found no change from 3 to 5 cm. Rieder and Eidinger[2] found that if

$$Hd^3 > 3 \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (2)$$

where H = Magnetic field strength, oersteds.
d = Electrode spacing, cm.

the arc velocity is independent of spacing between 3 mm and 3 cm. The tests described here have shown that in some parts of the range there is considerable change in velocity with variation of electrode spacing, although $Hd^3 \gg 3$, especially near the electrodes. This is probably because magnetism plays a large part in cathode movement, so that the criterion in eqn. (2) for non-ferrous metals does not hold for ferromagnetic electrodes.

The present tests have shown that, although over a certain range the cathode velocity is very similar on copper, brass, aluminium and zinc, there are differences at currents over 200 amp, and carbon and the ferromagnetic materials show velocities differing as much as 30 : 1 for the same current, and if carbon and pitted-steel rods are compared, the velocity ratio is 550 : 1. Because previous investigators have not tested ferromagnetic electrodes, and have not tested a number of non-magnetic materials at such high currents, these considerable variations of velocity with electrode material have not been discovered hitherto. Rieder and Eidinger[2] found no difference of velocity on copper, silver and tungsten, and Winsor and Lee[1] found no difference in regular cathode movement on silver, copper, molybdenum, tungsten, nickel and carbon. Babakov[3] stated that the arc velocity is determined by displacement of the cathode spot, since velocity depends on electrode material and oxide layers, but he did not give the changes in velocity.

Rieder and Eidinger[2] found that arc velocity, v, varied with current, I, and external transverse magnetic field, H, according to

$$v = CI^{0.61}H^{0.74} \quad . \quad . \quad . \quad . \quad . \quad . \quad (3)$$

where C is a constant. They suggest that this confirms theoretical explanations that the arc behaves like a solid-rod conductor moved through the air by a force HI against air resistance proportional to v^2. In view of the evidence shown here that arc movement is governed by transfer of cathode emission, it hardly seems reasonable to carry the simple rod-conductor idea to the extent of attempting to explain quantitative results.

Further evidence that the movement of an a.c. arc depends to a considerable extent on cathode emission transfer was obtained by measuring the average arc velocity on copper electrodes with a spacing of 1¼ in. The results are shown in Fig. 2, and it can be seen that, as with steel, an approximately linear variation is obtained, with velocities lower by a factor of 1·8. From Figs. 6 and 9 it can be seen that at low currents the cathode velocity on copper is about one-third to one-fifth of that on steel, but the velocities differ rather less at higher currents, so that the factor of 1·8 shows the great effect of cathode movement on the overall motion of an a.c. arc. The relation between average velocity, current and spacing given in eqn. (1) appears to hold over a very wide range. Tests were made by Freiberger[5] on copper busbars 250 cm apart with alternating currents up to 15 000 amp and his results are shown in Fig. 10. Calculating steel velocities from

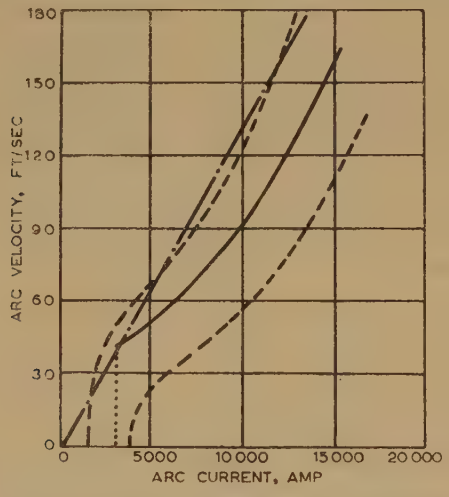


Fig. 10.—Comparison between calculated results and experimental results of Freiberger.

— — — — Limiting curves of Freiberger.
———— Mean curve of Freiberger.
— · — · — Line calculated from eqn. (1).

eqn. (1) and reducing them by the factor of 1·8 mentioned above to obtain velocities for copper, gives the line shown in Fig. 10 for a spacing of 250 cm. The agreement with Freiberger's results in a range of current and spacing well outside that tested here is remarkably good, especially since Freiberger's results are averages of a number of conditions, e.g. with barriers in the arc path.

It has been shown that the physical picture which is commonly employed of magnetically deflected arcs behaving as solid-rod conductors moving against air resistance is quite unrealistic and inadequate. It is clear that further work is needed to clarify the processes of transfer of cathode emission and the critical role they play in arc movement. Further experiments towards this end are desirable, where greater control can be exercised over such factors as electrode-surface irregularities and layers.

(7) ACKNOWLEDGMENTS

The work was carried out in the Short-Circuit Laboratory at Queen Mary College, University of London. The authors wish to thank Professor W. J. John for the facilities provided and for his help and encouragement. They also thank Dr. T. J. Lewis for valuable discussions and advice, and Mr. R. J. Williams for his assistance with some of the experimental work. Some of this work was done as a part of an investigation made for the Electrical and Allied Industries Research Association into the protection of high-voltage insulators from arc damage, and this part of the work will be described in E.R.A. Report Ref. O/T19. The authors wish to thank the Association for the use of the wind tunnel and camera.

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ABBREVIATIONS

(P)—Address, lecture or paper.
(p)—Subject dealt with in a paper or address.
(D)—Discussion on a paper.
(A)—Abstract of paper or address.

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A.C. network analyser. (*See* Network.)
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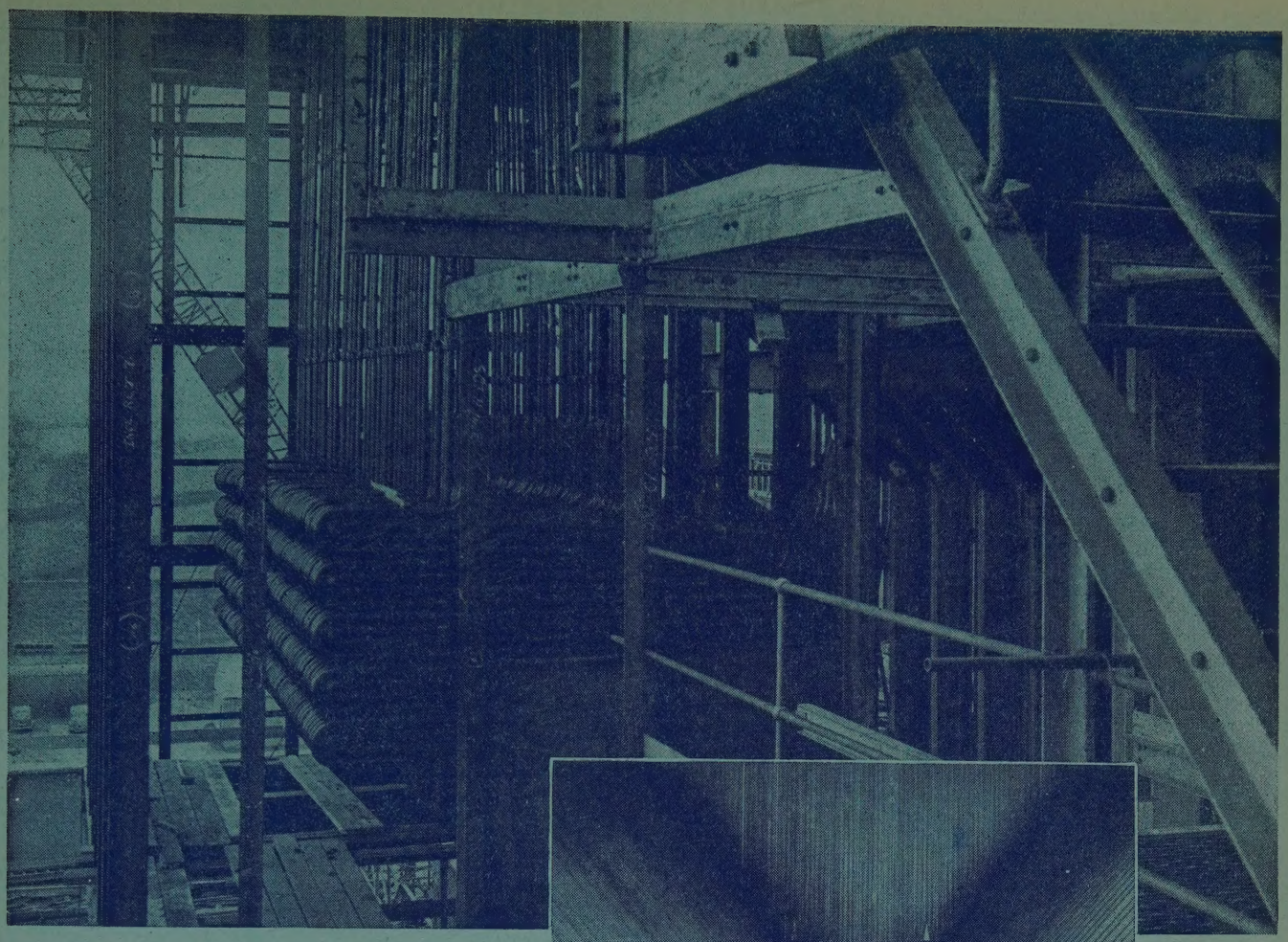
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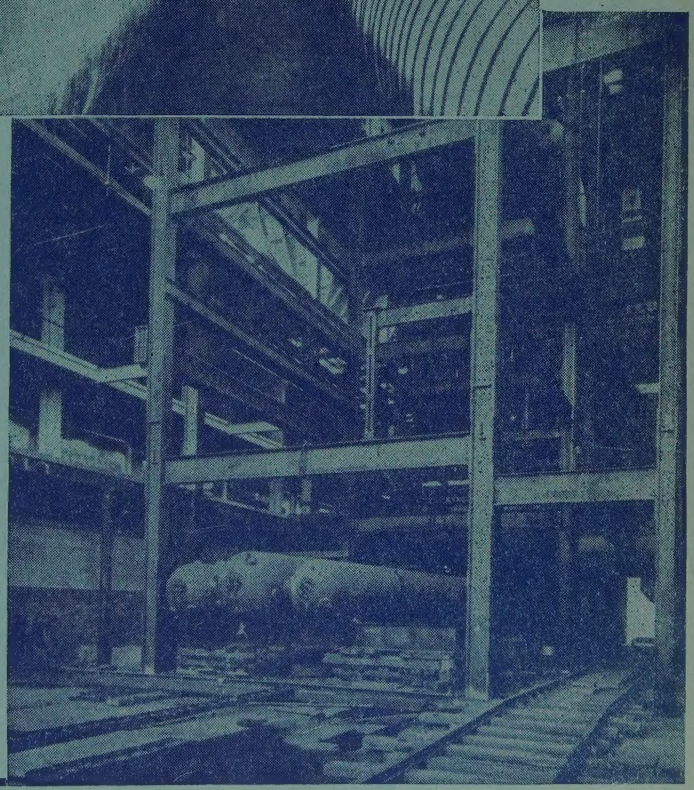
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Printed by Unwin Brothers Limited, Woking and London.